The MINING CONGRESS JOURNAL

COAL CONVENTION

Containing the papers presented to the Eighth Annual Convention of Practical Operating Men of The American Mining Congress.

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Contents

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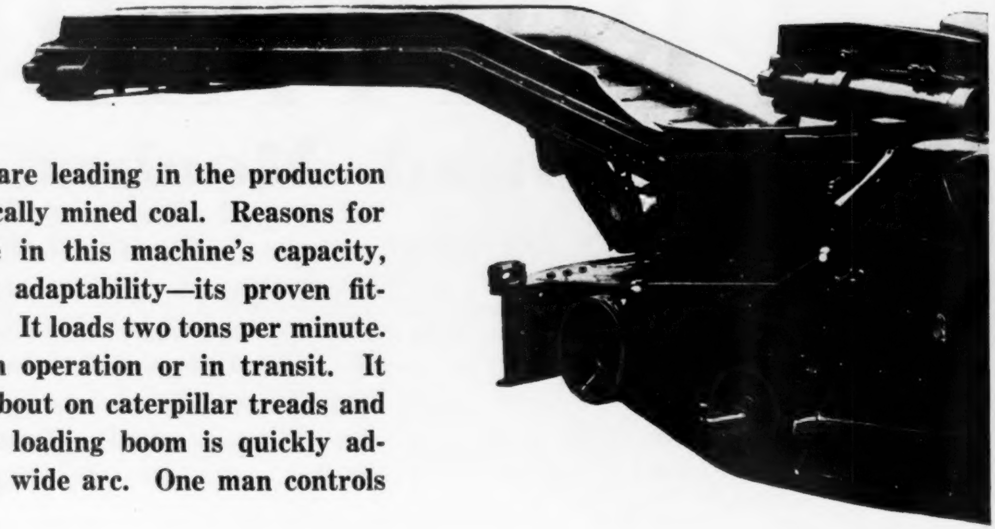
THE
5-BU



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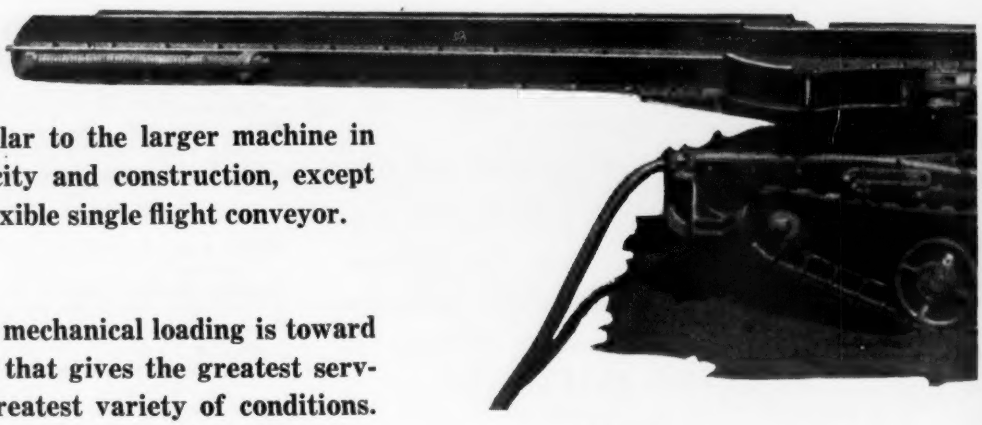

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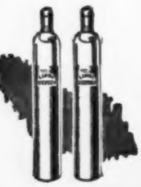

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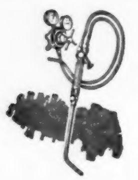

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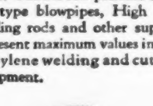

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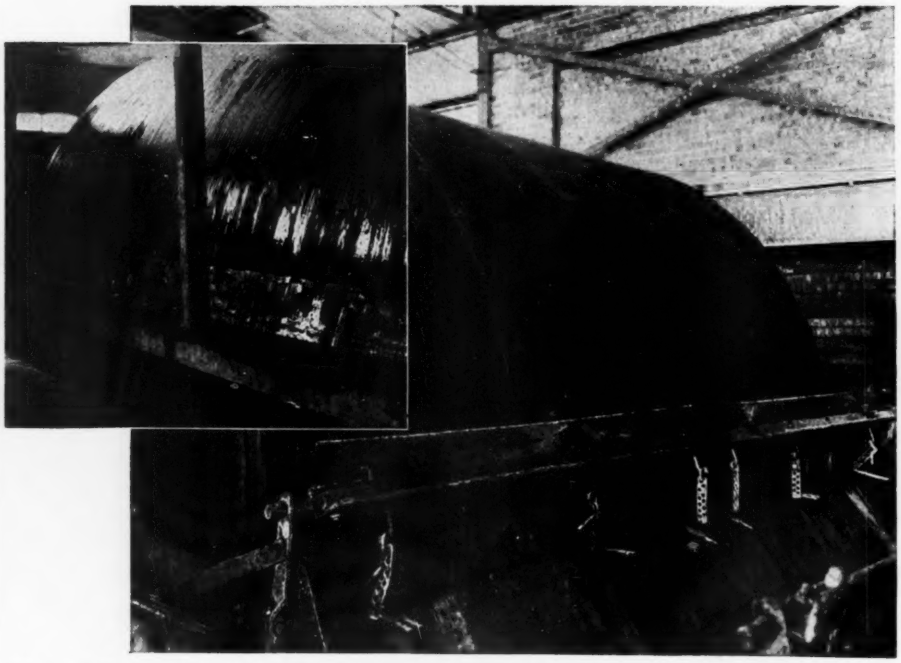

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The
MINING CONGRESS JOURNAL

A Monthly Magazine—The Spokesman For The Mining Industry—
Published By The American Mining Congress

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Acknowledgment

Incorporating the most advanced thought to date on the subject of coal mining and preparation, the 35 papers which appear in this special issue of The Mining Congress Journal are presented herewith solely through the efforts, and by the courtesy, of the American Mining Congress.

They were prepared especially for this year's annual American Mining Congress Coal Convention, and there they were first introduced and discussed.

The American Mining Congress has now authorized their publication in the present volume that their service may be permanently extended, not only to the benefit of those who attended this year's Convention, but to that of the entire Coal Industry as well.

E. R. Coombes
EDITOR

Trends Toward
Better Mine Management

By P. C. Thomas*

IN the United States there always have been some well-managed bituminous coal mines, yet even these have fallen in line with those that were not operated as well as they might have been and have bettered their methods. There is no need to screen the fact that up to, say, 10 years ago, although much money was made in coal mining and coking, the properties were largely in the hands of superintendents and foremen. These men probably did their best, but as loaded cars were their main object, little mattered so long as their tally was high. Managers visited the workings infrequently, and vice presidents and presidents bothered little with details—profits were their principal worry; how and at what cost were of less importance.

Coal mining always lagged behind metal mining in management, and comparisons were often made regarding this; also as to methods and the class of technical supervision employed.

However, during the past 10 years, particularly the second half of this period, a steady and great change has been apparent to those who have been observant of conditions in the coal industry. Companies of standing work fewer mines, install the best equipment, employ the best talent as supervisors and workers as well as cater to their welfare, and everyone—from the president down—is awake to the necessity of the best performance of man and machine.

The closing down and abandoning of small and unprofitable mines and the concentration of mining in a few large units has been and is a prominent trend in recent operations, and we have examples of companies that once worked four mines now have only one, and more than 50 reduced to 20. These fewer workings can produce an equal or greater tonnage of coal at less cost.

Inasmuch as the American Mining Congress represents the coal-mining industry as a whole, and the writer is familiar only with certain fields in the eastern part of the United States, to make this paper representative he sent a copy of a questionnaire to 25 well-known company officials in 14 states, as follows:

Alabama, two; Arkansas, one; Colorado, one; Illinois, three; Indiana, two; Kentucky, one; Missouri, two; New Mexico, one; Ohio, three; Oklahoma, one; Pennsylvania, three; Utah, two; West Virginia, two; Wyoming, one.

Most of the mines owned by these companies are large coal producers and have modern equipment and supervision. Replies were received from 20 of these men, and 12 states are represented; Arkansas and Missouri are not included in the replies. The total production represented by these replies amounted to 60,000,000 tons in 1929, which is 12 percent of the bituminous coal output of the United States for that year. Thanks are offered at this point for the prompt answers to the questionnaire, which were all-important to this paper.

The Questionnaire and Answers

The questionnaire consisted of the following items, followed by a summary of the answers and expanded opinion:

STAFF POSITIONS

1. Are vacancies in official positions filled from within your own organization or do you fill them from the outside for the purpose of injecting new blood?

Nineteen answers were "yes" or mainly so. If suitable men are available within the organization, they are promoted. There are times when a management has to go outside its staff for a particular man. Two companies endeavor to find the man best qualified without particular preference for insiders or outsiders. After experience of appointing men from within and without, one company became confirmed to the "within our own organization" policy. When mechanization of loading began, a departure was made from this for a while, but it was soon found advantageous to develop the experience and hold to the established and loyal men on the pay roll. Such promotion gives minor officials an incentive to be ambitious and excel in their work, but employment of outsiders discourages them. One company has this record: All vacancies have been filled within our organization with the exception of two general superintendents who were chosen in the past eight years; we anticipate our ability to take care of all future demands from our men trained on the property.

VALUE OF TRAINED MEN

2. Do you make a practice of employing technically trained men, and if so, in what capacities?

Everyone is agreed on this, but not all for all positions. These trained men occupy places on the staff from engineers and general superintendents to firebosses. One company employs them in all important capacities; another excepts superintendents and foremen. At one mine the young engineers are given the opportunity to follow any natural talent which they possess. A special effort is made by one company to place the sons of employes who have a combination of practical mining experience and are technically trained, and it looks forward to a future organization consisting of many of such men. One firm employs technically trained men in its engineering department only. Each year one group of mines adds a number of college graduates to its staff. They are started on time-study work, without authority or responsibility, and their advancement is subject to the development of their practical ability. One large operator encourages the procuring of technically trained men who are transferred from time to time to broaden their training. Superintendents and foremen frequently rise from the ranks, and they are well fitted to handle labor better than others. One company maintains a scholarship in mine engineering for some positions.

With regard to employment of mining engineers (not necessarily confined to coal mining), in the West they have been to a great extent responsible for elimination of the rule-of-thumb method, for many years a drawback to economics in coal mining.

COMPANY LOYALTY

3. What policy, if any, do you pursue to foster loyalty to the company among your men?

In brief, close contact between officials and men, a square deal, promotion, financial assistance, relief plan, good wages, group insurance, pensions, free information, and a genuine interest in the welfare of the men fosters loyalty. If their confidence is won, a company can expect the best of them. On the other hand, the management of a group of three companies reports limited success in instilling loyalty. One operator tries to make every man feel that he is an important part of its organization and that his advancement depends upon merits of his work. An industrial relations department whose prime responsibility is to establish relations between employer and employe, based on mutual interest, confidence, cooperation, and benefits is found to be of value in promoting loyalty. So is the publication of

* Vice President, The Koppers Coal Company, Pittsburgh, Pa.

a monthly safety paper which promotes logical reasoning and loyalty to the industry. A prominent Alabaman thinks that one of the forces that has aided the trend of better management is the closer contact between higher officials of companies and their employes. An Oklahoman thinks this way also, particularly since the virtual passing of the miner's organization.

COMPANY MEETINGS

4. Do you have periodic meetings of department heads and operating officials, if so, how often, and who attend?

All of the operators answered in the affirmative, although one stated that instead of periodic meetings, the heads are in daily contact with their organization. The frequency of these meetings varies from daily to quarterly and semiannually and, when convenient, and those who attend range from the president to the mine foreman, or salaried employes, or department heads, or all who can be present. One western company holds a monthly review of conditions attended by operating officials headed by a vice president. One plan is to hold monthly meetings, one of which includes all department heads and another of miners and company men; the first is designated the foreman's committee and the latter the employes' committee. All questions pertaining to safety and efficiency are discussed. In a group of 20 mines, superintendents gather once a month and staff officials weekly. For the plant and division meetings of one operator, the procedure or what is to be considered is carefully planned in advance with the idea of getting the best thought, at the same time leaving unity of action in the minds of local officials. In sum, meetings are considered to be essential to the working of any property.

MINE DEVELOPMENT

5. Are your mines being developed according to some definite projection plan? What steps are taken to insure the plans being followed and whose responsibility is it?

As would be expected from such a group of coal producers, all develop their mines on a definite plan. To insure this being done there is frequent inspection and surveys, checking, and posting of maps. The responsibility rests with the vice president, management in general, chief engineer, general manager, company chief mine inspector, or superintendent. Section bosses at one mine must follow the directions given by the engineering department. At another the production department is responsible for holding to the layout. All entries in one group are driven and all rooms turned on sights which are given by the surveyor, when required; pillars are drawn according to a projection and a schedule. At a western group, when unexpected conditions arise, the engineers develop new plans for approval by officials. Development at an eastern group is carried on in conformity with forecast maps which are prepared by division engineers, in conjunction with the chief engineer and division manager, and approved by staff members, and the general manager, of operations. Sometimes difficulty is encountered in trying to adhere to the plans.

METHODS OF OBTAINING BEST RESULTS

6. Do you employ an efficiency engineer? If so, to whom does he report and what are his duties; and if not, what effort, if any, is made to obtain efficiency such as increased tons per loader, cars per locomotive, and other operations?

Only one of 20 operators employs a regular efficiency engineer; another does so occasionally; the remainder depend on their chief engineers, managers of mines, and assistants—getting the best results is part of their work. These engineers report to the vice president; sometimes to the president. An operator in the central field writes that each department head is his own efficiency engineer, and he is expected to maintain or increase the efficiency of the details in his department. He often places men for time-studies or particular observation when necessary. In the production engineering division of the engineering department of one large property the duties include those of the so-called efficiency engineer as well as many other duties—planning, estimating, scheduling, budgeting, and so forth. As to getting the best results in loading and hauling, there are daily reports of tons per loader and a monthly report of cars per locomotive; competition between superintendents and foremen; miners are supplied with plenty of coal and cars and motors operate on a prepared schedule; time-studies; constant effort; and responsibility of mining engineer, electrical engineer, and maintenance engineer.

The question of haulage has received increased attention, particularly in the better application of power, larger units, larger mine cars, better track, resulting in cheaper transportation. Haulage equipment and tracks in one mine in New Mexico are maintained in the best possible condition consistent with economy.

In Ohio, one company gives constant attention to haulage in the matter of power, larger mine cars, and better track.

Haulage costs in one Utah mine have been lowered materially by having 4 to 7-ton cars with roller bearings, 56-lb. rails, small gathering locomotives, and long trips on main haulageways.

EQUIPMENT STANDARDIZATION

4. What methods are followed to have standardization of equipment?

Methods vary in this phase of mine operation, but more or less standardization is maintained or an atteempt is made to do so as far as possible or where possible. It is rather difficult at some mines to standardize, particularly as so many new machines are being developed, and at others the standards are revised occasionally. One method is to check all purchases to have equipment and spares as uniform as possible. When obsolete plant is to be replaced at one mine, standardization starts with the new equipment. This problem becomes a thought of operating heads and is frequently discussed at staff meetings. The cooperation of superintendents leads to more or less standard methods. Interchangeable equipment in a group of mines was the practice by one company, but the purchase of other mines somewhat upset this arrangement. However, in hand-mechanical loading only one type of machine is used, and in power-mechanical loading, because of the advance in machines, there had to be a little departure from the standard. The standardization of equipment does prevent the carrying of excessive stocks and repairs. If a company operates in several fields, local conditions do prevent complete standardization; but when in the same district the problem is relatively simple. In fine, uniformity of equipment is desirable and essential, within practical limits.

A Pennsylvania president admits that mechanization of mines has been progressing, but all mines do not lend themselves to mechanical loading. With a gradual lowering of the wage-scale and only a limited running time, the financial advantages of mechanization in many fields have become non-existent. The only justification for the capital expenditure necessary for mechanizing a mine is a market which will take the product of that mine running a reasonable number of days a week. With the current wage-scales in non-union fields where a mine is not operating over three days a week, the economy of machine loading is doubted.

A Utah manager is of the opinion that where their use is feasible, mechanical loading machines have been main factors in reducing costs at coal mines.

ACCIDENT PREVENTION AND WELFARE

8. Do you have a separate department in charge of an individual to look after accident-prevention work? (a) Do you have safety inspectors, and if so, how many does each inspector look after? (b) Do they have any other duties besides inspecting for unsafe conditions and practices? (c) Do you have 100 per cent first-aid training? (d) What is done, if anything, in the way of following up the activities of company doctors, public health matters, and smaller allied activities?

Six companies do not have such a department, 13 companies do, and one company considers that all are concerned in its safety work. The department may be in charge of a casualty manager, or of a trained safety engineer, whose duty it is to train men, maintain the rescue equipment, and take care of all accident and compensation cases. The practice of inspection varies—there are safety inspectors, foremen, and bosses, also, committees representing the labor union. There may be an inspector for one mine or for a group of two to four mines, and he may look after 30, 50, 100, 800 or 3,500 men. Most of these inspectors haven't any other duties. One policy regarding inspectors is to have a few men to each boss who has full authority and responsibility for safety and output. In the case of the mine with one inspector for 3,500 men, the company has many trained miners and therefore accidents have decreased; when inexperienced miners are employed more inspectors are required. In the foregoing it was remarked that at one mine all are concerned in safety work; the company does not believe in a separate department or individual to look after accident-prevention work, because it is a matter of such importance that from the management down it should be considered along

with all major operating and selling features.

As to complete or 100 per cent training, half of the operators have had their men instructed in first aid, some of them try to, and some are nearly fully trained; the remainder have not. On January 1, 1931, a southern company established a rule requiring men to have first-aid training before they will be given a job, also that 90 days is given to those already employed to acquire such training.

With regard to following up the activities of company doctors, public health, and allied work, some operators watch the results carefully, others have the doctors report, others have the chief surgeon check the work, whereas others don't give much attention to it beyond what the doctors and public health officials do. In one case an experienced manager is assisted by labor representatives. Another producer finds that if company doctors look after sanitary conditions and the health of the men and their families, their morale is improved. Two others make regular inspection of company houses and surroundings and samples of domestic water. Another relies on general supervision by officials and superintendents. Contagious and other diseases card records from doctors in its mine villages are received by one company which follows up public health matters through a welfare nurse, welfare worker, and sanitary engineer. One division of one company maintains a fully equipped hospital.

The coal industry is constantly giving increased thought and effort to its accident-prevention work, with definite and profitable results. At one group in Alabama, matters are left less to the judgment of the worker, particularly in timbering, which has fixed rules and standards devised by engineers.

A great deal of constantly increasing thought and effort have been given to accident-prevention work in Ohio. and this should show definite and profitable results.

One Utah company makes every effort to promote safety within the mine and to instruct employes to observe safety rules so that danger may be seen, recognized, and avoided. It costs 2½ to 3 cents per ton of coal for this work.

MAINTENANCE OF DISCIPLINE

9. What methods of discipline are applied to insure observation of safety rules?

Apart from a central State operator whose methods depend on the support given by the Labor Union, discipline at the other properties is maintained by the usual reprimands, fines, layoffs or suspensions, demotions, and final discharge. Flagrant violations or one that involves serious consequences may result in discharge for the first offense. Three companies give each man a book of rules and expect him to study them and act accordingly. Competition for safety records between mines helps, so does cooperation between company and men. One corporation has the practice of making each boss who has an accident to one of his men appear before his superintendent to explain why; the superintendent has previously received a detailed report of the occurrence from the safety department. Foremen and face bosses are subject to the same discipline as miners and others if they permit unsafe conditions to exist.

COSTS

10. Do you have a separate Cost Department? Do you work up detailed costs for each section or only for each time, and to whom is this information given?

As would be expected, nearly all of these operators have a cost department, although this work may be done in the accounting or auditing department. The average practice is as follows: Detailed costs are worked up for each section, mine (mostly the mine), division, or department, and the information is given to all heads, general superintendents, mine superintendents, foremen, or to all concerned or to those who are interested. Daily cost sheets are compiled for each mine of three groups, available the next day. As to costs for each section, one producer finds this unnecessary save when some special information is desired, another prepared a monthly cost sheet for each section and for each individual foreman, showing both labor (100 men to each foreman) and supplies. An interchange of costs between officials of all divisions is in vogue by one corporation. Another large producer in its operating department has a cost accountant who has the title of assistant chief of mine clerks. He supervises the preparation of cost data for each day period. Another system is cost accounting by jobs rather than by geographical sections.

A manager writes from New Mexico that their costs are kept in considerable detail, so that at the end of each month the different items can be compared with previous months and any irregularities called to the attention of those directly responsible and thus corrected.

An Ohio president believes that there is a definite trend toward better cost accounting, but there is still much room for improvement. It is known that some companies deliberately omit certain items of expense to show low cost during periods of low prices.

A president in Oklahoma states that the income tax and resultant checking and auditing of returns has brought about a great reform in mine accounting and compelled operators, who previously were indifferent, to employ auditors and bookkeepers who have put in proper systems of cost accounting. Continuance of fierce and increasing competition has so reduced prices and profits as to compel better mine management.

Cost accounting is necessary, one president in Pennsylvania answers; any reasonably intelligent operator must realize that; but when operating figures are persistently in red, it is a great temptation to leave out some of the cost figures. There is also a temptation to get work done on capital account instead of being placed against the cost of coal. That sort of thing has to be watched with special care when cost is higher than realization.

In Utah, a general manager answers, mine costs in detail are compiled by accounting departments at each of the mines, and these are forwarded to a central accounting department which handles the whole business. Costs are easily compared. Costs were reduced by two mines which now yield as much or more than five mines did formerly.

A West Virginia official states that his observation has been that while there has undoubtedly been a trend toward better cost accounting and more efficient management on the part of some of the larger companies, yet in the case of many, especially the smaller ones, it would appear that there has been little or no improvement along this line, particularly in knowing what is their actual production cost. Although there has been improvement in management, preparation, and other operations, there seems to have been little or no improvement in marketing methods, and this is where there is room for drastic change.

PURCHASES

11. Do you have a centralized purchasing department or does each mine buy its own supplies locally? Do you have a central store-house, and what practices are followed to secure minimum inventory in storehouse?

All of the companies approached on this question replied in the affirmative; but with regard to having a central storehouse, half do and half do not. At three groups there is a supply-house at each mine, and at two groups there is an interchange of materials, a practice in vogue at properties other than those concerned in this questionnaire. With regard to inventory, there are card systems, a daily balance of material on hand, or continuous inventory, and inventory once a month or several times a year. Two firms hold the purchasing departments responsible for minimum stocks. Another curtails buying when it is found that supplies are excessive. Supplies at several mines are purchased only on requisitions approved by the purchasing agent, superintendent, chief electrician, or master mechanic; and at one mine a daily record of supplies issued is given to the supervising officials and management.

INSPECTION OF COAL

12. Do you have coal inspectors, and if so, do they inspect at the face or at the tipple?

Excepting one mine with a daily output of 900 tons, all of the producers have coal inspectors, all at the tipple and several at both the face and tipple. Occasionally inspection is made at the face by one company, but this is done by underground men selected for the job. Another company's inspectors see that miners load clean coal, also as it is being dumped and as it goes over the picking tables. There is one practice of an inspector watching the coal as it is being dumped by a rotary dump at the bottom of the shaft to determine at which working face dirty coal might be loaded. All inspection is done at the working faces of one group of captive mines. Coal inspectors representing the sales department of one company inspect the coal on the tipples and occasionally at the face. At another

(*Continued on page 6*)

Budgeting Repair Costs

By B. H. McCrackin*

IN PRESENTING this paper we will confine ourselves to the general study of an outline of the methods used to control, to a large extent, the expenditure necessary to keep in good operating condition all mechanical and electrical equipment, exclusive of mine tracks, etc., which is used in the production of coal at an average mine.

The cost of maintaining the electrical and mechanical equipment, at any given mine, in which such equipment is used to any great extent, is a substantial part of the total cost of producing coal, and with the trend toward mechanized mining, this part of the cost assumes more and more importance.

Much time and study has been devoted to working out a budget system; that is, projecting ahead and controlling within specified limits the cost of many of the basic activities which enter into the production of a ton of coal, but, considering its importance, a relatively small amount of study has been devoted to budgeting the cost of maintaining the equipment necessary to produce that ton of coal.

It is generally accepted among coal operators that the cost of repairs can not be controlled within close limits, and that due to the ever-present possibility of the failure of some piece of equipment the cost of repairs will be great or small depending upon the frequency and magnitude of such failures. It must be admitted that the failure of mechanical or electrical equipment will occur and any system of controlling such costs must necessarily recognize the possibility of such failures and allow for them under the heading of emergencies. It is evident that the success of such a system depends upon reducing such emergencies to a minimum and the practical elimination of these failures or emergencies must be effected through preventive maintenance; that is, a large part of the money spent for maintenance of equipment must be made on a basis of prevention of failures rather than from consideration of rebuilding after a failure has occurred.

Repair costs have a tendency to fluctuate, and if they are to be controlled it is evident that the high points in such a cost curve must be leveled off and the abnormally low points of this same curve must be built up with such expenditures as are necessary, but which are definitely controlled. This method of expenditure may seem contrary to the generally accepted policy of maintaining equipment under which no expenditures for repairs is made unless absolutely necessary, but it is contended that before repair costs can be kept at an economic level, all expenditures for such work must be definitely controlled. To repeat, maintenance costs can only be controlled by eliminating breakdowns, and to eliminate them the expenditures must be made from a standpoint of prevention. After the breakdowns have been largely eliminated, it will be possible to budget and reasonably control the cost of repairs.

There are two factors which comprise repair costs: first, labor; second, material. A study of the relation of these two factors at any mine will prove very interesting. Normally the material item is greater than that of labor, and any study towards reducing the total cost must necessarily deal largely with the material item. It is a fact, however, that the relation of the material cost to the total repair cost is largely dependent upon the labor component and the reduction and stabilization of the total cost must be attained largely through the labor item. The labor item is directly under the control of the mine management and an attempt is often made to reduce repair cost by cutting off repairmen when a reduction in mine cost is desired. It is a fact, however, that when the labor component in the repair cost is decreased beyond an economic point, the material component immediately increases far beyond the decrease in labor, making the total cost higher than before the labor reduction. This statement is, of course, based on maintaining equipment in good condition and not deferring maintenance as is sometimes done.

To prevent mechanical or electrical failures, equipment must be inspected at regularly stated intervals and faulty conditions removed when found and parts, worn sufficiently, replaced. Our problem, therefore, resolves itself into determining the amount of the labor item at any given mine which, properly supervised in inspecting and reconditioning equipment, will reduce the material component to a minimum. With the establishment of this labor cost and its resultant material component we have determined the maintenance cost for this mine on the basis of prevention of failures. This cost may be considered as the normal repair cost and projected ahead as such.

Preventive maintenance is largely a problem of inspection of equipment at such intervals as may be necessary, taking into consideration the importance and usage of the equipment to be inspected and allowing sufficient time during such inspection to make replacements of parts which are sufficiently worn. Each and every piece of equipment in use must be inspected at such intervals as have been predetermined and such adjustments and replacements made as are found necessary. Thus it will be seen that repairmen become, primarily, inspectors trained to detect and remove the possibility of failure before it actually occurs. Such a system trains repairmen to become keenly observant and always on the alert to detect conditions which may result in failure.

Daily reports submitted by each inspector serve to show what he has accomplished during his shift, and also provides a running record of repairs made to the equipment inspected.

The amount of time to be allotted for inspections and the frequency of such inspections depends upon the kind and type of equipment, and becomes a problem to be solved for each classification of such equipment at every mine. For instance, a short-wall mining machine requires one hour for inspection and repair each day or operation; a track cutting machine from two to three hours for the same period; a gathering locomotive one and one-half to two hours depending on whether it is an open or "sealed" type; a haulage locomotive from one to one and one-half hours depending on whether it is engaged in light or heavy service. An electric mine fan normally requires two hours per week and a steam mine fan three to five hours in the same period. Motors, conveyors, and other tipple equipment require a thorough inspection daily, the amount of time depending on the size, with the average 2,000-ton tipple requiring one man per shift. Cages, ropes, caging equipment, etc., must be thoroughly inspected once each shift.

From the foregoing it is evident that the problem of determining the normal labor item of repair cost for any mine becomes one of adding up the time required for the inspection of all equipment which, if a failure occurred, would interrupt the production of coal, and once this labor cost has been determined it becomes a simple matter to project it over any given period.

It might be interesting for the purpose of illustration to assume a mine with the following equipment in use and calculate the labor normally necessary to maintain this equipment economically. The mine has a 3,000-ton output per eight-hour shift, the mine opening is of the drift type with the haulage locomotives delivering the coal to the tipple. The equipment consists of a shaker screen tipple, 600 3½-ton steel mine cars of the end-dump type and equipped with tapered roller bearings, eight 10-ton haulage type locomotives (one of which is a spare), fourteen 6-ton gathering type locomotives (one of which is a spare), eight track cutting machines (one of which is a spare), and approximately 60 field or gathering pumps ranging in size from 50 to 250 gallon capacity for low head operation.

The tipple will require for the purpose of daily inspection and adjustments one mechanic working an eight-hour shift. To obtain the maximum efficiency from this man it would be advisable to stagger his working shift as compared with the tipple operating time; that is, his shift should start two or three hours after the tipple starts, and would continue a cor-

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responding length of time after the tipple has finished the day's operation.

For maintaining the mine cars, two men each working eight hours will be required. One of these men should be a blacksmith due to the all-steel construction of the cars.

The main line locomotives will each require one and one-half hours of inspection and repair per 24, or a total of 12 hours per day. This figure includes enough time for periodic retrucking and overhauling.

The gathering locomotives will require two hours inspection and repair per 24, making a total of 28 hours. This figure includes enough labor for periodic retrucking and overhauling.

Each track cutting machine will require two hours per 24, or a total of 16 hours. This figure includes enough time for periodic overhauling.

The problem of determining the amount of labor necessary to maintain the gathering pumps in this mine must be based entirely upon assumption. If the water to be handled is non-acid and the regular pumpers are required to lubricate the pumps under their supervision, one man working an eight-hour shift would be sufficient to take care of them. However, since the average mine has more or less acid water to handle we will assume that this condition exists, and the number of men required to maintain the number of pumps in operation will probably be about three, each working an eight-hour shift.

In addition to the above, one general blacksmith with a helper, each working an eight-hour shift, will be required to take care of miscellaneous smithing and tool sharpening.

The figures given are based on an average. Hard-cutting and difficult locomotive hauls will increase the amount of time necessary to maintain the cutting machines and locomotives, while extremely favorable conditions will slightly reduce the figures shown.

The determination of the material item in the repair cost is not so simple nor accurate as the labor. The only means of obtaining it is through records over some period long enough to be of value, say at least a year. The longer the system has been in force the more accurate the average figures will be to project ahead, giving due consideration to those projects which it is known will have to be started or completed during the period over which the cost is to be projected. Such average figures, however, will be found to be accurate within all reasonable limits.

The success or failure of any maintenance system depends entirely upon the mine organization. To facilitate the functioning of the preventive maintenance system the usual mine organization is changed slightly. The ordinary chief electrician and master mechanic positions being combined into one position known as the maintenance foreman, who works directly under the mine superintendent, and on a par with the mine foreman. The duties of this position are many and varied. He is primarily responsible for maintaining all electrical and mechanical equipment in good operating condition and therefore directs all inspection and maintenance activities at the mine. All inspector-repairmen, mine car repairmen, blacksmiths, etc., are under his direct supervision, as are also those men who operate and to some extent care for stationary equipment such as hoisting engineers, fan engineers, boiler firemen, etc. He writes all orders for supplies to be used for repair work and keeps all the time for maintenance activities. He sees that all necessary reports are properly made out as required by the men under his supervision, countersigning such where indicated.

In addition to his direct maintenance duties, he studies the operation of the various kinds of equipment to determine whether it is being properly operated, since equipment which is not properly operated can not be economically maintained. Where improper operation is found, leading to abuse or inefficiency, it is his duty to take up with the proper authorities and have such misoperation discontinued, as this will have a very direct bearing on maintenance cost.

This one man has at all times a full knowledge of the condition of all equipment at the mine and from this knowledge is able to plan the whole maintenance program. If normal costs have been low during a month, he can overhaul some piece of equipment such as a mining machine or a locomotive or some other necessary expenditure which was scheduled for the next month. It is possible for the maintenance foreman to so schedule all his work that he will maintain practically a level cost curve. When a breakdown occurs it is treated as an emergency and the increase in cost, if any, explained as such. Projected costs, however, normally have a small allowance for slight fluctuations in labor and material and will cover the infrequent breakdowns.

The "preventive maintenance" system has many advantages over the "repair-after-the-breakdown" system, some of which are as follows:

(a) Practical elimination of equipment failures with the resultant loss in tonnage and interference with scheduled activities.

(b) Repair costs can be budgeted within reasonable limits.

(c) The cost of maintaining equipment is less, for it costs less to keep equipment in good condition than to allow it to break down and then rebuild.

(d) Repair work can be scheduled, for with the complete knowledge of the equipment which a close inspection system gives, periodic overhauling can be definitely foreseen, and a definite schedule set up.

(e) There is a tendency to develop a better class of repairmen. The inspection system develops the ability to observe closely for those conditions which may cause failures.

(f) The inspection system makes it possible to check each inspector's work closely. Failures may be classified as lack of proper inspection or unavoidable, and a record of breakdowns and their cause readily exposes the careless or inefficient workman.

Trends Toward Better Mine Management

(*Continued from page* 4)

mine the foreman inspects the faces, and at a Western mine the tipple boss.

The preparation of coal has gone forward in spite of low prices, and both the producer and the consumer know more about coal as a fuel today than they did several years ago.

One producer in New Mexico finds that it pays to give considerable attention to preparation, and men are employed to inspect carefully each railroad car of coal after it has been loaded.

In Ohio, the preparation of coal has been and is receiving considerable attention.

One Pennsylvania president tersely sums up coal preparation thus: "Coal is being better prepared; it has to be to sell at all."

One general manager in Utah feels that perhaps the mines in general have gone too far in the preparation of so many sizes of coal.

Conclusions

The whole purpose of this trend toward better mine management is to mine better coal, uniformly clean, and prepared for market with a minimum of accidents to employes and at the lowest possible cost, at the same time keeping our properties in the best physical condition and treating our employes as well as we can. Staff positions are mostly filled by men from within the organizations. Technically trained men are employed for most openings. Various systems are in force to foster loyalty to the companies. There are either periodic meetings of staff men or there is close contact among them. All mines are developed and maintained on a definite layout or plan. Evidently efficiency engineers are not popular, because they are not employed by this group of operators who expect efficiency from within. If possible, standard equipment is used or equipment is standardized for the particular group of mines. The majority have an organized accident-prevention department, although they expect a certain amount of what might be termed natural safety within the force. Discipline is enforced, as it should be, by warnings, suspensions, and dismissals. Cost accounting receives considerable attention, as do purchases. And coal is carefully inspected before it leaves the mine for the consumer.

Maintaining Discipline

By Thos. G. Fear*

TO A great many people the word "discipline" is quite unpleasant, and usually recalls memories of penalties incurred for doing the things they desired to do, but which, according to some higher authority, was not the proper procedure.

Discipline is one of the very fundamentals of our civilization, and is simply our mode of living according to instructions. From the time of the Veeda, Koran and Bible to our present city, state and Federal laws human beings have had instructions to guide and direct them in their actions.

Discipline, as related to coal mining, therefore, requires that those in authority formulate certain instructions pertaining to our mode of living in and about the mines, and in order that these instructions shall be the proper instructions, it is absolutely necessary that the persons formulating the instructions are themselves amenable to discipline, and also have the proper experience.

The executives and supervisory forces should keep in mind and obey the present day Ten Commandments which have been assembled by Col. M. C. Rorty, vice president of the International Telephone & Telegraph Company:

1. Definite and clean-cut responsibilities should be assigned to each executive.

2. Responsibility should always be coupled with corresponding authority.

3. No change should be made in the scope of responsibilities of a position without a definite understanding to that effect on the part of all persons concerned.

4. No officer or employe occupying a single position in the organization should be subject to definite orders from more than one source.

5. Orders should never be given to subordinates over the head of a responsible officer. Rather than do this the officer in question should be supplanted.

6. Criticisms of subordinates should, whenever possible, be made privately, and in no case should a subordinate be criticized in the presence of officers or employes of equal or lower rank.

7. No disputes or differences between officers or employes as to authority or responsibilities should be considered too trivial for prompt and careful adjudication.

8. Promotions, wage changes and disciplinary action should always be approved by the officer immediately superior to the one directly responsible.

9. No officer or employe should ever be required, or expected, to be at the same time an assistant to, and critic of, another.

10. Any officer whose work is subject to regular inspection should, whenever practicable, receive the assistance and facilities necessary to enable him to maintain an independent check of the quality of his work.

Successful business managements have been using these instructions for years and the concise manner in which Colonel Rorty has laid down these Ten Commandments should impress them indelibly upon our minds.

With the supervisory forces in the proper frame of mind to enforce discipline, the next requisite is the formulation of the instructions or rules.

This matter must be handled very carefully and each department should be requested to draw up tentative rules covering the department.

These department rules must next be combined, and all conflicting rules eliminated after which the resultant set of rules should be passed on to each department for correction and ratification.

Our own experience in formulating a set of safety rules for 47 mines operating in ten different coal seams in four different states demonstrated the advisability of this method of procedure. Rules set up which are not enforceable detract from the degree of enforcement of good rules.

The best example of this is the disregard of many state and federal laws since the passage of one Volstead law.

Education—The next step after the rules have been made is to educate the men so that they may all have a working knowledge of what is expected from them. Our plan is to have schools for our supervisory forces and when these men have passed a satisfactory examination they in turn instruct the men in their departments.

Penalties—A uniform system of penalties must be set up to insure fair and impartial treatment to all violators of rules although it is impossible to fix the penalty for each and every offense. Our system provides for a warning for the first offense; two days' suspension for the second offense; ten days' suspension for the third offense, and discharge for the fourth offense. For flagrant cases of violations of the safety or coal preparation codes, the offender may be discharged for the first offense when in the judgment of the mine superintendent the violation was premeditated and intentional.

All the officials of the company are subject to these same rules and superintendents and mine foremen have been suspended and in some cases discharged for violating the rules. Not even the president and vice president are exempt as the records of our Safety Courts show fines have been collected from both.

When the offense is noted by a member of the supervisory force, a discipline report is immediately made out and forwarded to the mine office.

A card index file on each employe is kept at each mine, and also in the divisional headquarters, where all violations and penalties are recorded. From this mine file the superintendent can determine the number of the offense which automatically sets the penalty, except in special cases.

Each offense is carried on the record for a period of 12 months, after which it is dropped and the record cleared.

It was our belief that in fairness to the employes, the records should be cleared periodically in order that slight infractions of the rules a year apart would not cause suspension or discharge.

Each employe has the right to appeal to the division personnel manager who investigates the case and if he believes the man was not treated properly, the entire case is referred to the division manager for settlement.

Fairness and firmness are the two essential requisites in a foreman or superintendent to enable them to mete out proper discipline, and many intelligent and honest men have failed in handling men because they lacked the necessary nerve to do the thing they knew ought to be done.

The success of a discipline system depends entirely on how it is received by all and, when the employes believe they are being treated fairly and without partiality and that they themselves can have the supervisory force disciplined through fines imposed by our Safety Courts, there is no resentment.

Cleanliness of the mining towns, plants and underground workings has a very decided effect upon discipline; where the mine management keeps the plants in a tidy and orderly manner, the men naturally form better habits. This same effect is also noted that where the safety discipline is strictly enforced, it is much easier to enforce preparation of coal rules or any other efficient operating rules.

To summarize this subject in a few words, set a good example, be fair and firm and you will get results.

* General Manager in Charge of Operations, Consolidation Coal Company.

Safety Program at ARMCO

By Chas W. Connor*

ON account of widely varying natural conditions and differences in operating methods no single set of rules and no one method of procedure can successfully govern the application of a safety program to mining operations in different fields, or even to individual operations in the same field. Each mine must of necessity adopt the type of program that will best suit its geographical location, mining conditions, methods of operation, and the class of labor employed. In discussing briefly some of the features of a program which has proven successful at an individual operation and in presenting some of the results which were accomplished, it should be stated here that this plan is not put forward as being infallible or perfect in every particular, nor is it recommended for adoption in its entirety by any other operation. Certain general principles of the plan can, however, be universally applied.

The mine at which this program is in effect is the Nellis Mine of the American Rolling Mill Company, located at Nellis, Boone County, W. Va., in the Kanawha mining district. This property was acquired by Armco in 1920, and development was begun at once in the No. 2 gas seam, which here maintains an average height of 53 inches. The annual production of the mine is normally about 400,000 tons, and the average number of employes is approximately 300 men.

The coal seam is overlaid with a heavy draw slate of varying thickness, which is very treacherous due to the presence of numerous slips and "kettlebottoms." On exposure to air this roof slate cuts along the ribs and spalls in the middle of working places, creating a serious hazard to the man at the working face. Most of our accidents are due to falls of roof slate, and a great part of our efforts have been exerted to reduce injuries from this source. These facts are mentioned here to show that whatever favorable results have been attained in accident reduction, have been achieved in spite of discouragingly poor natural conditions.

For several years after the acquisition of the Nellis property, as the demand for tonnage was increased, accidents likewise increased both in number and severity, and compensation rates mounted rapidly. On account of these conditions our general management decided to extend to the coal mines the same accident prevention program which was already in effect at our steel plants. It was about the middle of 1926 that the Safety and Training Department first began active work at Nellis.

We were fortunate in that we did not have to sell our executives on the value of safety. Armco was one of the pioneers in the safety movement and for many years had carried on this work. They knew its value both from the humanitarian viewpoint and in actual dollars and cents.

Charles R. Hook, president of the company, was already on record with the statement that: "No man can hope to go far in this organization who does not regard safety as one of his major responsibilities."

He had also said, "There are many things in connection with my duties which I am compelled to pass on to my assistants, but the record of every accident occurring in our plants comes over my desk and I personally read and study the record."

With the background established by these statements it was, of course, not at all difficult for the Safety Department to secure the active cooperation of the Mines Department, and, with the efforts of both departments directed toward the same end, it was not long before the safety program was initiated and in full swing.

Let me say here that no program of this kind will be successful which does not have the full sympathy and backing of company executives. In any undertaking for safety and accident prevention where executives are not fully sold on the value of the work, the task of convincing them should become the first objective of the program.

It was perhaps not quite so simple and easy to get started off on our work as the preceding statements may seem to imply. In the first place it was imperative that we get started off "on the right foot." It was intended that this should not be a campaign of the evangelistic type, starting off with lots of "whoopee" and then suffering the slow, lingering death common to its kind, but a permanent program with definite aims and objectives.

It was considered of prime importance that whatever was done should be done in such a manner as to impress on our employes the fact that we were thoroughly in earnest and that the safety program was here to stay. We believed that we, as well as the workmen, had a duty to perform, and that we must show a willingness to do our part before we could consistently ask for their cooperation.

With this thought in mind a comprehensive survey of the entire plant, both inside and outside, was made for the purpose of determining every existing or potential hazard, sub-standard condition, and dangerous operating practice. When this survey was completed and the unsafe conditions catalogued, we started systematically to remedy or eliminate them. This work was not, by any means, accomplished all at once. It proceeded gradually and in an orderly manner, and with as little interference as possible to regular operations.

In some instances the remedy for a dangerous condition was quite obvious. Others were more involved and sometimes quite expensive. The most glaring dangers were corrected first the others followed. It is not necessary to enumerate all the things we did. Suffice it to say that we "cleaned house thoroughly. Loose slate was taken down; timbers were set; tracks were overhauled; airways were cleaned up; traveling ways were provided; guards were installed; safety equipment was purchased and put into service.

All of these improvements were visible to the employes, or at least they had common knowledge of what was being done. And all the while they were, unconsciously perhaps, imbibing the new safety atmosphere — becoming safety minded.

When our work had progressed to the stage where we felt we could come before our men with "clean hands," and ask them to join with us in the prevention of accidents, we did so and their response was indeed gratifying and inspiring. Since that time we have had practically 100 per cent cooperation from our men. Occasionally we encounter an individual who can not or will not carry out instructions. We lose little time in getting rid of a man of this kind, and in releasing him we tell him plainly why we do not care to work him.

Our work was further simplified through our contact with an organization of employes known as the Nellis Armco Association. Principally formed for beneficial purposes in case of death, or non-compensable sickness and injury this association formed an ideal means of contact between the local management and employes, and rendered valuable assistance in promoting interest in our safety work.

* Superintendent of Mines, The American Rolling Mill Company, Nellis, W. Va.

	Total Accidents	Tons Produced	Man Hrs. Worked	Tons per Accident	Man Hrs. per Accident	Days lost on Acct. of Injuries	Frequency Rate	Severity Rate
1926	56	321,622	623,280	5,743	11,130	6,735	89.90	10.81
1927	24	361,047	696,553	15,044	29,023	13,576	34.50	19.50
1928	20	368,622	717,433	18,431	35,872	13,989	27.90	19.50
1929	17	389,428	708,833	22,908	41,696	11,644	23.97	16.42
1930	7	303,318	553,683	43,331	79,071	505	12.65	.91

Some idea of the effectiveness of our program may be gathered from the accompanying statistics, covering the period from 1926 to 1930, inclusive.

The statistics given are accurate records. Man hours of exposure is not a matter of conjecture or estimate since all employes are checked in and out on a time clock. Days lost and frequency and severity rates are calculated on the same basis as that used by the United States Bureau of Mines in the national safety competition.

A major accident is defined as—"any accident causing a man to be incapacitated for his regular work for more than the remainder of the day, turn or shift on which the injury was received, including any case resulting in death within two years of the date of the injury, also compensation cases in which the injured man loses no time from work"

We have been asked whether the decreased number of accidents in 1930 might not have been due to reduced working forces and working time, incident to lessened production in that year. Reference to the column in the foregoing table showing man hours of exposure per accident effectively disposes of that question.

A comparison of accidents during the five-year period shows for 1930 a reduction below previous years as follows:

1929	58.82	percent
1928	65.00	percent
1927	70.92	percent
1926	87.50	percent

Seven calendar months of 1930 were "no-accident" months.

The period from July 22, 1930, to the end of that year (162 days) was free from lost time accidents, and this record was continued to January 19, 1931, making a total of 180 consecutive days without a single lost time injury.

Our compensation insurance rate was reduced from $4.20 per hundred dollars of pay roll to $1.88 per hundred dollars, the present minimum rate in West Virginia. In this item alone we have made a saving of approximately $12,000 per year.

We are frequently asked the question: "What has been the most important factor in bringing about the reduction of accidents?"

Frankly, we would hesitate to attribute this result of our work to any one line of endeavor. Very early in our safety work we learned that production and safety go hand in hand. We learned that when a job was done right from the operating standpoint it was also a safe job. When we improved conditions for safety purposes, we automatically secured greater efficiency from both men and equipment. Production per man hour was increased and costs were decreased, notwithstanding that we were doing many things that were not formerly undertaken. We found that every dollar spent for safety was directly or indirectly contributing toward better operating conditions. We, therefore, stopped regarding the two as separate problems and attacked them as one. In doing this we directed our efforts along five different lines which will be briefly described under the following heads: (1) selection and placement, (2) education, (3) training, (4) supervision, (5) discipline.

Selection and Placement

Our early experience with men who would not comply with our rules, and with men who were not physically fitted for their tasks soon convinced us of the necessity of selecting our workmen and of placing them in jobs for which they were best suited. The most desirable type of men from a safety standpoint are those who are intelligent, healthy, able-bodied and amenable to discipline. In order to secure this class of labor, we have tried to make our community attractive to men of this type and to their families by providing surroundings, which they will appreciate, such as proper treatment, good wages, comfortable houses, schools, churches, bath and change houses and a merchandise store supplying goods to employes on a "no-profit" basis. Recreation facilities are provided and sports of various kinds are fostered. Free life insurance, in an amount equal to one-half of one year's anticipated wage or salary, but in no case in less amount than $1,000 is provided for every employe who has been in the continuous service of the company for one year or more. Additional contributory insurance may be purchased by employes at low rates. Eighty-five percent of our mine employes take advantage of the contributory insurance privilege.

All men undergo a physical examination by the company surgeon and each is given a rating which defines the type of work he is physically capable of performing. His proper placement is determined both by this rating and his previous experience. We have no hard and fast rule as to age limit, but actually we do not employ men over 45 years of age unless their physical condition and ability are exceptional.

We employ no colored labor. Men of foreign birth comprise about 17 percent of our entire force, the remaining 83 percent being native born white Americans.

Selection and placement of men has unquestionably played an important part in building up a working force much above the average in intelligence and physical make-up. It has also tended to build up a more permanent group, which is most desirable. Labor turnover at Nellis is very low. In both of these respects, the results of our work along this line have been favorably reflected in our accident records.

Proper selection and placement of men is worthy of consideration in any company's safety program.

Education

Education and training are generally considered under the same heading. We differentiate them because in our practice they are really separate and distinct subjects.

The education of our workmen commences the minute he is assigned to a job. He is immediately given a copy of the state mining law and company rules. Before going to work he is instructed in regard to the hazards of his job, his tools are inspected to see that they meet requirements, and he is told to whom he will report. On entering the mine his instruction is continued by the foreman, who advises him of safety regulations and explains our standards and standard practices.

Opportunity to improve himself educationally is afforded through weekly classes. The first of these classes is conducted in cooperation with our local schools, and are held at night. These classes cover elementary subjects and are extremely helpful to those lacking early educational advantages.

Mining classes are held in cooperation with the Mining Extension Department of West Virginia University. These classes appeal to the men with aspirations for better jobs and are well attended.

Classes in safety and first aid and subjects allied to mining are a part of the curriculum of the local schools. The results of this teaching gets back to the home, and at the same time makes the children of the community our best safety boosters.

Monthly safety meetings are helpful both educationally and in the way of maintaining interest. These meetings are open to the workmen and their families. Higher officials of the company are frequently present. An interesting program is planned for each meeting. Community singing, local talent entertainments and music, and free movies are presented. Business conditions are briefly discussed. Problems peculiar to our own plant are talked over. Men prominent in the mining industry or in other walks of life are invited to make safety talks. These meetings have served a useful purpose in our safety work in furnishing an additional means of contact and making safety a live subject at all times, and they are always given a crowded house.

Attractive bulletin boards are provided, both inside and outside the mine, on which are displayed safety posters, news pictures, accident reports and statistics, foremen's records, letters and notices of general interest.

A small plant paper called "The Safety Post" is issued monthly, usually at the time of the monthly safety meeting. In this publication safety subjects, correct practices and items of local interest are presented.

All of these things and many not mentioned contribute directly to the safety education of the workman and are helpful in securing his support.

Training

In 1929, believing that we had gone as far as possible with the ordinary educational methods, Armco initiated and developed a program of "foreman-manager training." This was undertaken because in our opinion the foreman is

really the keyman, the man who must be depended upon to bring about results. It was our thought that he could work more efficiently and get quicker and better results, if he had a broader and more intimate picture of his responsibilities. In other words we wanted to develop in him the managerial viewpoint. The principal aim of this course was to train the supervisory forces, so that they in turn could train the workers. Safety was not forgotten in outlining this course. On the contrary it was strongly emphasized, but it was regarded as only one of the component parts of the program—not as a thing separate and apart.

One of the most commendable features of the course was its utter simplicity. It was not a text-book program. Blanket outlines were prepared and distributed by the Safety and Training Department. The course covered six months with one month devoted to the study of each of the following main heads: Labor; machinery and equipment; material and supplies; accidents and illness wastes; steam, oil, electricity, air, water, etc.; cleanliness and order.

Weekly meetings were held and methods of overcoming the operating difficulties presented by the outlines were suggested and discussed. The fourth meeting of each month was always devoted to a discussion of costs of the general subject being covered. It would take too long to tell of all the details of this work. Many valuable suggestions were offered, many important decisions made, which were to favorably affect our future work. The supervision learned more of what it was all about and how to get things done in the proper way.

The next logical step was the training of the workers. While the foreman-manager course was continued with a broadened scope, job training classes were started for the benefit of the various classified labor groups. These classes were conducted by our own supervisory forces. Meetings were held with each labor group including loaders, machine men, motor crews, trackmen, drillers, slatemen, timbermen, pumpers, wiremen, electricians, tipple crews, etc. Classes were continued with each group until they had been fully informed as to the correct way to do their job. If you have ever watched a half dozen different trackmen do their work you will know that they do it in a half dozen different ways. They may all be good ways, too. But there is bound to be one way that is best, or at least a best method may be worked out by combining the best features of each. That is what was done in job-training our men. After a full discussion and often with the aid of models or sketches, a decision was arrived at as to the best way of doing a job, from the standpoint of safety, efficiency, and cost. This method then became our standard practice for that job, and no other method was permitted. Practically every operating job at Nellis has been so standardized, and practical rules have been adopted for the government and guidance of men in the various classifications. Job training is simply instructing the worker in the proper technique of his job and its purpose is to teach the average worker to become an expert worker. As an example of what may be accomplished, let me refer to our motor crews. Haulage is usually regarded as one of the hazardous occupations in coal mining and the haulage system is commonly productive of numerous injuries. Since job-training our motor crews there has not been a single lost time accident on our haulage system in over 23 months. Tipple and washery crews have a record of more than 3½ years without a lost time injury.

Once we stopped regarding safety as something apart from production, and began to train our men to do their jobs right, our accident experience began to improve by leaps and bounds.

Every man at Nellis has been job-trained. The classes are carried on anew each year with each group to insure their being familiar with standard practices and to permit of changes that may, from our experience, have been proven necessary.

The effort involved in this training is considerable, but the results provide ample justification. We have found that it pays and recommend it as an effective means to reduce accident frequency.

First aid and mine rescue training are also conducted. The purpose and results of this type of training are generally understood and will not be discussed. We believe in the value of work of this character. We believe in it so strongly that first-aid training is compulsory at Nellis and our men are 100 percent first-aid trained. Employes are trained by our own instructors and are given an Armco certificate on completion of the course. Teams are also trained which compete in plant, interplant, district and statewide contests.

Numerous instances occurring in our own plants proving the value of the emergency treatment of injuries have convinced us that first aid is entitled to a prominent place in our safety activities. Without doubt the moral effect of this training tends to make men more careful to avoid injury.

Supervision

Responsibility for carrying out the safety program rests upon the general superintendents. This responsibility is, of course, delegated down the line to the foremen. All our inside foremen are certificated men, selected from the working forces from our own plant. This system of promotion from within our own ranks is adhered to wherever possible and provides an incentive for the man who is trying to move a step further up the ladder. The new foreman, when promoted, is already familiar with our policies and methods and no "breaking-in" period is necessary. No foreman has more than 30 men under his charge, giving him a group which he can take care of effectively. To the foremen is delegated the responsibility for production from his section, and for the safety of the men under his care. It follows naturally that, having this responsibility, he should also have the necessary authority to maintain discipline. In our practice he is clothed with this authority. Small monthly bonuses are paid to foremen having clear accident records of more than 30 days.

In addition to supervision by the plant officials, inspections are made at regular intervals by our own Safety Department. Reports of these inspections together with accompanying recommendations reach not only the operating officials but members of the general management as well. Inspections are also made by the state department of mines. It is our purpose not merely to comply with legal requirements but to go beyond these requirements in keeping abreast of modern practice in safety and operation. We use safety lamps, rock-dust our mines, provide rescue stations and equipment, and do many other things, not because we are required by law to do so, but because we believe them to be most effective safety devices and in keeping with the practice of any company which has real safety and the prevention of accidents as its goal. Close and effective supervision is essential in any well-planned safety program.

Discipline

In beginning our safety program considerable thought was given to discipline and methods of enforcement. Safety courts, mine safety committees, and other means of forcing compliance to rules, were studied. Some were tried out and one by one discarded. We found that penalizing an offender by small cash fines, public reprimands, or reports by inspection committees of his fellow workmen were not conducive of the best results, and often left a rankling suspicion in the mind of the offender that he had not been justly dealt with.

After all is said and done, the person most vitally interested in the prevention of injuries is the individual foreman. His evidence in connection with any accident is usually paramount. He has the closest contact with his men, and has the most intimate knowledge of their behavior and the conditions under which they work. He has pride in his record, his bonus provides an excellent reason for permitting no laxity, and he knows that his continued employment depends in great degree upon his accident performance.

After several experiments, we decided that we could get rid of complaints and cut out the red tape by placing responsibility for discipline squarely on the shoulders of the foreman. We have concentrated our efforts, therefore, on securing foremen of the proper caliber.

We have been slow to adopt written rules and regulations. Such as we have are few in number and are rigidly enforced. These rules are incorporated in our standard practices, and the foremen use these for their guidance. An infringement of these rules usually means a reprimand from the foreman for the first offense, suspension for second offense, and dismissal for a third offense. The nature of the violation sometimes varies the application of the punishment, but no constant offender is long tolerated on any job. This plan might not work in all localities, but it has given us better results than any other we have tried out, and our experience impels the assertion that given the necessary authority, we believe the foremen will, as much as any other factor, determine the success or failure of a safety program.

In conclusion let me say that whatever success we have had at Nellis has been accomplished through careful study and planning, and has been brought about by the efforts of a working organization that has acted as a unit after a decision has been reached.

Today we say, "There is no job at Nellis mine at which a man *has* to get hurt."

PENNSYLVANIA MINING SYSTEMS

By M. D. Cooper*

THE object of this paper is to discuss four typical mining systems in present-day use. It may provide a means of comparison with former systems, or with others now in use elsewhere, or it may furnish a starting point from which systems better adapted to local conditions may be developed.

It is of interest to find that mining plans are now being laid out with greater attention to future retreat operations rather than to immediate questions of size of pillars for proper roof support. In other words, pillars of certain minimum size may be sufficient in strength to protect haulage roads, but they may be altogether too small to provide economical working sections on retreat. Longer range considerations and more intelligent provision for the future have resulted in larger barrier pillars.

Figure 1 shows the method of open-end mining by hand loading. The method is not new, having been in use for several years. Development consists of 32-ft. center butt entries 10 ft. wide and 300 ft. apart. Rooms are on 112-ft. centers with chutes on 75-ft. centers between rooms. Both rooms and chutes are driven 12 ft. wide, while cut-overs are 15 ft. wide. Blocks or ribs of coal 63 by 100 ft. resulting from this development are mined by the open-end system of mining.

Development is made on face and butt courses regardless of grades, as the coal bed has a uniform face and butt and works best on these courses. Drainage is made to conform with the mining system; ditches are provided to conduct water by gravity to low points where field pumps are located. Grades on haulage roads are also ignored in the desire for rapid rib extraction which, for single-shift operation, is at the rate of 120 ft. per month on face course. Approximately 24 percent of the coal is mined in development, leaving 76 percent for retreat. Average recovery of the entire seam for mineable height is 95 percent. Development is done by track machines with 9-ft. cutter bar, the coal being undercut and sheared in each place.

The machines used in development are also used in retreat, which consists of driving face cut-overs 15 ft. wide along the gob until the block of coal is nearly square, when cut-overs are driven along the gob on both face and butt courses until the block of coal is completely mined out. Blocks of coal are made oblong in the beginning to increase the number of working places per 112-ft. room. The number is 3.8 places, including development. When coal is undercut and sheared, two holes with four sticks of powder (6 oz.) per hole are used. Of the cuts, 63 percent can be sheared, while 100 percent can be undercut.

For timbering, three cribs are used at the entrance to the cut-over. Cribs are built of blocks each 5 by 6 by 30 in. A minimum of one cross bar per cut and one crib every two cuts with two rows of posts on 18-in. centers between cribs is standard for normal conditions. Additional cross bars and cribs are used as needed to meet roof conditions. Timbering is done as loading of coal progresses. Working places are made safe for the miner regardless of the amount of timber required. The miner sets posts as a part of his work, but is paid piecework for all cross bars and cribs.

Falls are made by a crew of four men, with the aid of a gathering locomotive. First a row of center, or safety, posts is set, one post per cut for the length of fall to be made. Timber recovery begins at the back end. All cribs, cross bars, and posts that can be taken out in safety are recovered, cross bars and posts being sacrificed where necessary to recover cribs. Recovery is approximately 90 percent of cribbing blocks, 80 percent of cross bars, and 45 percent of posts. The miner does not assist in making falls. Largest falls are 75 by 15 ft., or 1,125 sq. ft. From two to four hours are required for making a fall.

Ventilation is conducted along the front of the rib line and returned back of the gob, keeping full pressure on the front of the gob at all times. To guarantee that an ample supply of air will pass the last working place, bleeders are regulated back of the gob. This arrangement sweeps the gob of all dangerous gases and allows polluted air to bleed directly into the return.

At every sixth room a double landing is provided, the output from six rooms being sufficient for at least two gathering motors. Landings are kept within a radius of 1,000 ft. from the rib line. From each 112-ft. room approximately 130 tons of coal are obtained.

All of the coal is hand loaded at present. However, satisfactory trials have been made with machine loading on this system. To load coal satisfactorily by machine, proper preparation is essential. This was obtained by top-cutting, undercutting, and shearing. Coal thus prepared was shot down by using two holes with two sticks of powder in each. It was also necessary to place a row of posts on 18-in. centers along the rib before the lateral fall was made, the posts preventing the gob from sliding into the prepared coal in the process of loading out the cut. Otherwise, the same system of timbering that applied to hand loading also applied to machine loading.

Figure 2 shows the master plan that is used by the second of the companies whose methods are presented in this description. Within the panel enclosed by main faces and main butts may be worked whatever system of mining is necessary. The number of entries and the entry detail have been designed for, first, ventilation requirements, and, second, transportation. *Figure 3* shows a panel of rooms that have been turned from both sides of the butt entry. The rooms are called wing rooms and are used for hand loaders. Except that the wing would not be carried, the plan would work equally well for loading machines. Generally, the plan is used where cover is light, approximately 100 ft. or less. Another plan followed by the same company makes use of a section that was developed for block rooms on 90-ft. centers. Rooms are not over 14 ft. wide, being held to that width on account of holding head coal. Sometimes wide rooms, from 28 to 30 ft., are used and the slate is taken down. Both sections were designed for full retreat work. Room centers are selected to suit local conditions, the general range being from 35 to 45 ft. When loading machines are used in any full retreat panel (except 90-ft. blocks), on account of more rapid extraction, it has generally been found desirable to decrease the room centers to a minimum of about 35 ft.

The width of the panel or the length of the butt entries ranges from 1,500 to 2,000 ft. The length of the panel is such that it will provide a working section of convenient size. Main or flanking entries are driven in fours or sixes to provide sufficient places for the loading machine, adequate ventilation, and side tracks. Where six main entries are driven and side tracks are used in the main faces, the additional number of entries tends to offset the obstruction to ventilation due to cars standing on side tracks. Also, where double track main haulage is needed the additional entries counteract the effect of haulage within the entries and decrease the total mine resistance. Side tracks were within the butt entries, whereas the side tracks now are moved out to the main face entries. The change was necessitated by the use of mechanical haulage in servicing not only loading machines but also hand loaders.

Within full retreat panels, pillar lines are maintained at an angle of approximately 45 degrees and a length of 1,700 to 2,400 ft. Room widths vary from 21, 24, 26, 28, 30 to 35 ft., while room centers range from 33, 35, 39, 45 to 50 ft. Rooms are driven generally to a depth of 300 ft. if loading machines are used and 255 to 275 ft. if loading is done by hand. Depths are determined upon by the length of time that a room will have to stand before its pillar will be brought back.

No change in practice in removing pillars is made on account of loading machines. Pillars are drawn by driving butt-off places from 15 to 18 ft. wide, leaving a pillar about 6 ft. wide which is recovered from within the butt-off

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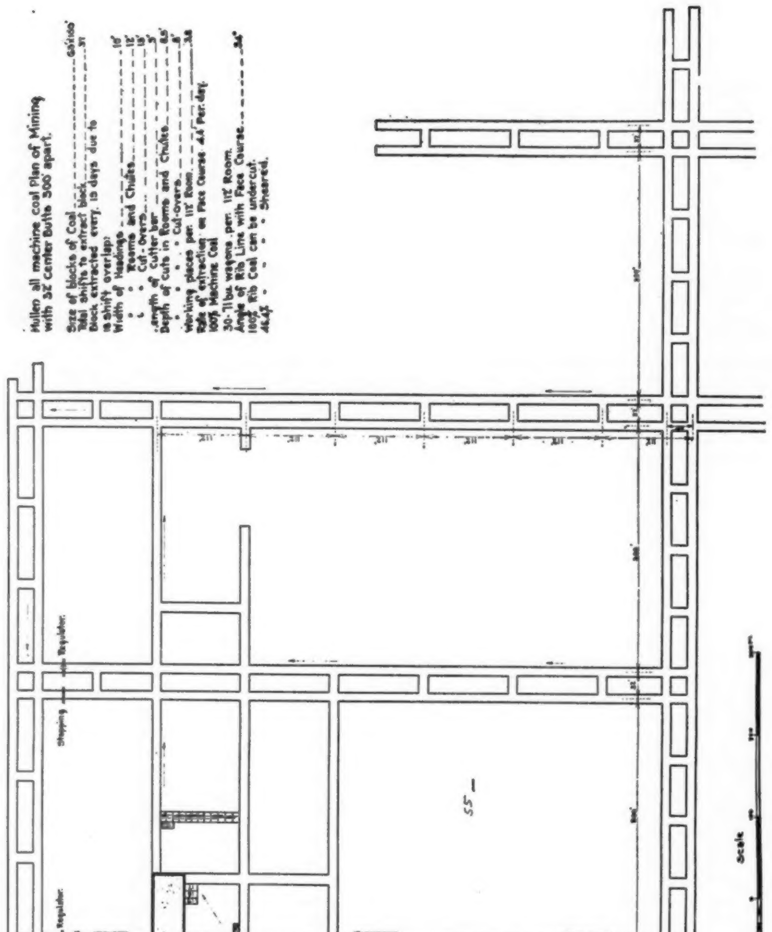

Returns
Stopping
Regulator
GOB
Scale
Mullen all machine coal Plan of Mining
with 32' Center Butts 300' apart.
Size of blocks of Coal 60'x100'
Total shifts to extract block 37
Block extracted every 15 days due to
18 shift overlap:
Width of Headings 10'
" " Rooms and Chutes 12'
" " Cut-Overs 18'
Length of Cutter bar 9'
Depth of Cuts in Rooms and Chutes 8.5'
" " " " Cut-Overs 8'
Working places per 112' Room 3.8
Rate of extraction on Face Course 44' Per day.
100% Machine Coal
30-71 bu. wagons per 112' Room.
Angle of Rib Line with Face Course 34°
100% Rib Coal can be undercut.
46.4% " " " " Sheared.

Figure 1

place. In mining wing pillars, a mining machine is used, but a protecting stump is left at each end and recovered by pick miners. When the last butt-off place is driven through and the recovery of the wing pillar between it and the gob is started, another butt-off place is begun. Where conditions are normal, yet another butt-off place may be started after the second one is holed half way through, providing three working places. The only difference between using hand loaders and mechanical loaders in a section like this is the more rapid speed attained by loading machines. Where the loading machine develops the rooms on a butt entry it may also mine the pillars and a sufficient number of places will be provided for one loading machine on one entry. It is possible to make a rib line 1,700 to 2,000 ft. in length provide work for three loading machines.

At another mine, in the Thick Freeport coal, the bed averages about 7 ft. in height of hard structure coal with a blackjack bone and bony parting varying from 12 to 14 in. near the center of the bed. The top is strong slate and sand rock which requires no timbering in rooms or entries. There is hard fireclay bottom. The coal bed is approximately level; cover is 200 to 300 ft.

Room and pillar system advancing with mechanical loading in rooms and entries is used. All pillars are recovered by mechanical loading.

Panels 1,380 ft. long are developed by four butt entries with 20 rooms on 65-ft. centers driven 350 ft. long off one side of the fourth entry. Four-butt entry system is used to give required number of places for the loader in advancing entries. All rooms are started off the butt entry at the same time. They are turned on 65-degree angles off the butt entry with room break-through turned on 60-degree angles on 72-ft. centers, as shown in *Figure 4.* Two loading machines are used in a panel, each machine working 10 rooms. Rooms are driven through to the next panel, which is on the retreat, and thus form the next retreating panel. The pillars are brought back by a crew on the pillar section. This system of mining eliminates the possibility of squeeze riding far as the panel is protected by solid coal. Ribs are extracted 100 percent; all stumps not loaded are shot out.

The loading machine working on development operates double shift with a 2½-hour intermission between shifts. Coal is prepared on the same shift, enough places being worked to keep ahead of the machine, which loads directly into mine cars of 5½-ton capacity placed under the loading boom one at a time by a gathering locomotive, one locomotive serving the machine. A single track of 30-lb. steel on 42-in. gauge is laid in each room. Track in each room is connected with the adjacent room by cross-over through the break-through. The gathering locomotive uses track in the adjacent break-through and room for shifting cars during the loading operation, hauling a trip of eight cars to and from the side track, which is a short haul.

All places are drilled by electric drill. Coal is cut 9 ft. in the center of the bed; the band of impurity is completely cut out. Then the place is sheared in the center of the face. The slack or bug dust is loaded out and the place is cleaned up before the face is shot down. Six holes are used in each place, which merely shatters the coal, giving a large percentage of lump. Permissible powder is used, being shot with electric detonator and battery.

In rib sections, a different type of load-

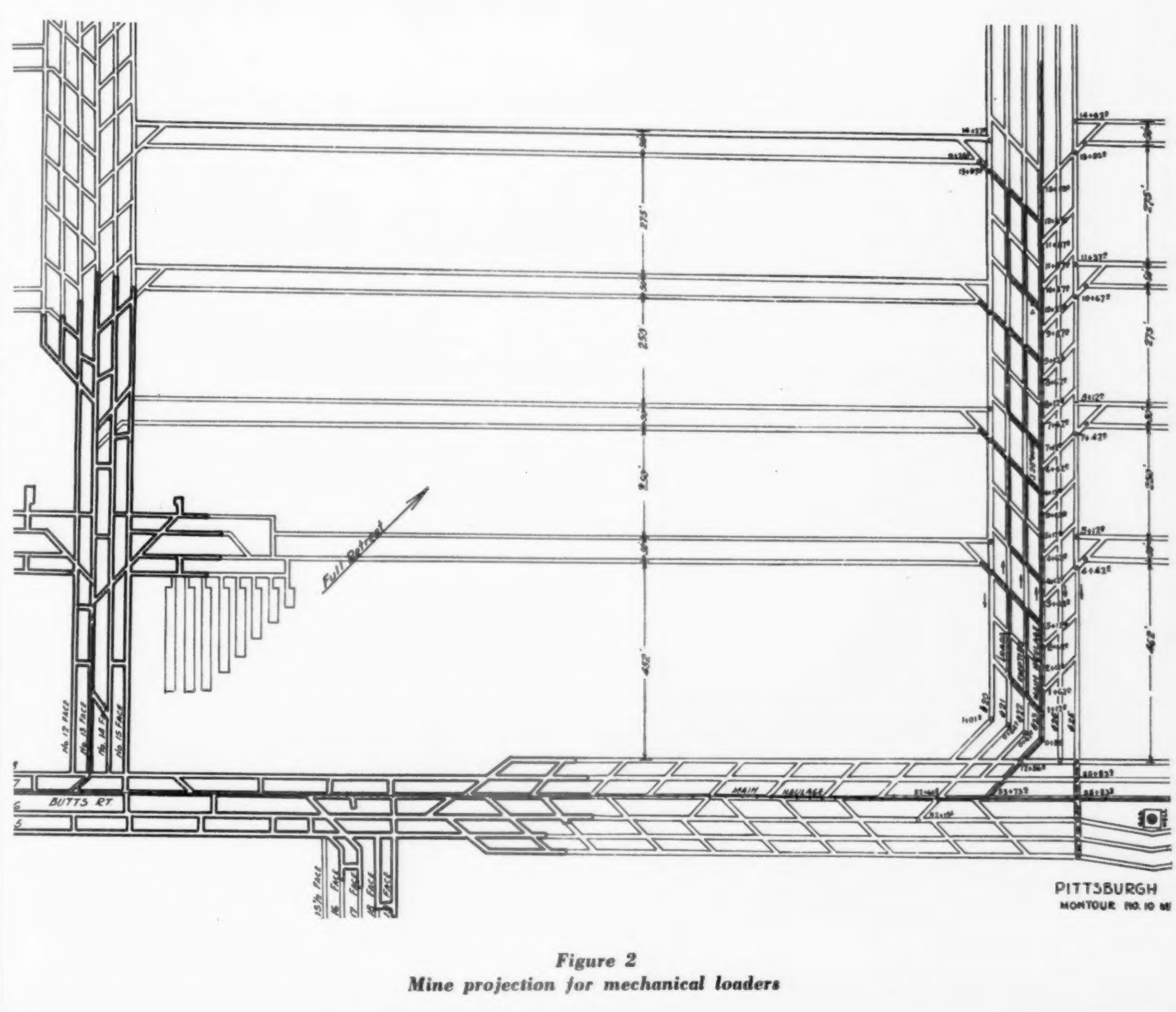


Figure 2
Mine projection for mechanical loaders

ing machine is in use. This machine has about twice the loading capacity of the machine in development work. It is served by two battery locomotives which are constantly shifting cars under its boom. Very little time is lost on car changes with the two-locomotive plan.

All preparation in rib sections is done on night shift and coal is loaded and hauled on day shift. The ribs are worked on open-end system and the machine loads along a face 55 ft. long, from which 120 tons of coal are loaded. The machine is 50 ft. in length, with three conveyors which, when filled to capacity, carry approximately 2½ tons with each revolution. A 5-ton car can be loaded in less than a minute. There are four machines of this type in operation and five of the smaller type. The small type are used mostly in development work and a sixth small type is kept in reserve in case of a breakdown.

Roof conditions are of such nature that very little timbering is required on advance work. With the exception of one small section, the mine is practically free from posts and timbers, which is of great advantage to mechanical loading. Wherever a timber is needed, hitchings are cut in both ribs and a steel rail is used as a cross bar to protect bad top, or top is shot down and loaded out. In the rib sections, under the present system of mining, it requires a large number of posts. The ribs are being worked on open end. Therefore, there must be ample protection to hold the face and safeguard the workmen. Posts are set in rows on 4-ft. centers, and parallel to the face of a working rib. They are also on 4-ft. centers at right angles to the fracture line. Posts are 8 and 10 in. in diameter with a wide cap piece set on top. Setting posts in this manner forms a triangle of 91 posts with 13 posts in the inside row at all times. Each cut, two rows consisting of 25 posts in all, must be pulled from the gob side of the triangle and the same posts reset on the working face side of the triangle, as shown in *Figure 5*. This requires labor in setting posts, but taking into consideration the amount of coal loaded from each face, it figures 25 posts set for 120 tons of coal loaded from each rib. The cover is light and this system of timbering has been successful in controlling roof.

In the rib sections, timbering is the first operation. Then the track is pushed over to the proper position and the electric drill, mounted on a truck which moves under its own power, drills the long face. The cutting machine then comes into the place and takes two cuts in the bony band, completely cutting the band out. Bony and bug dust are loaded out and the place is ready for the shotfirer. Coal is shot down and ready for a loading

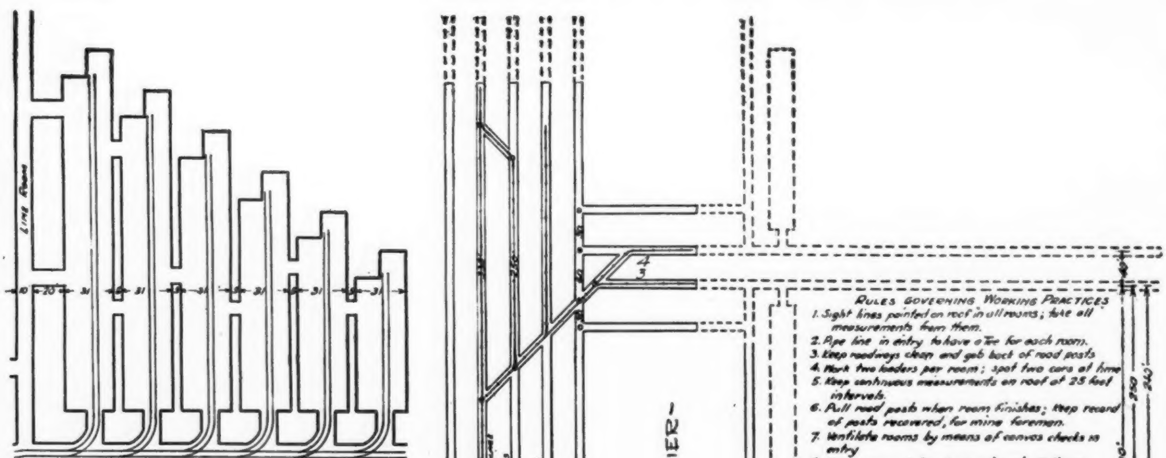


Figure 3

Method of working full advancing with wing rooms. For use in mines with shallow cover; no pillar mining attempted

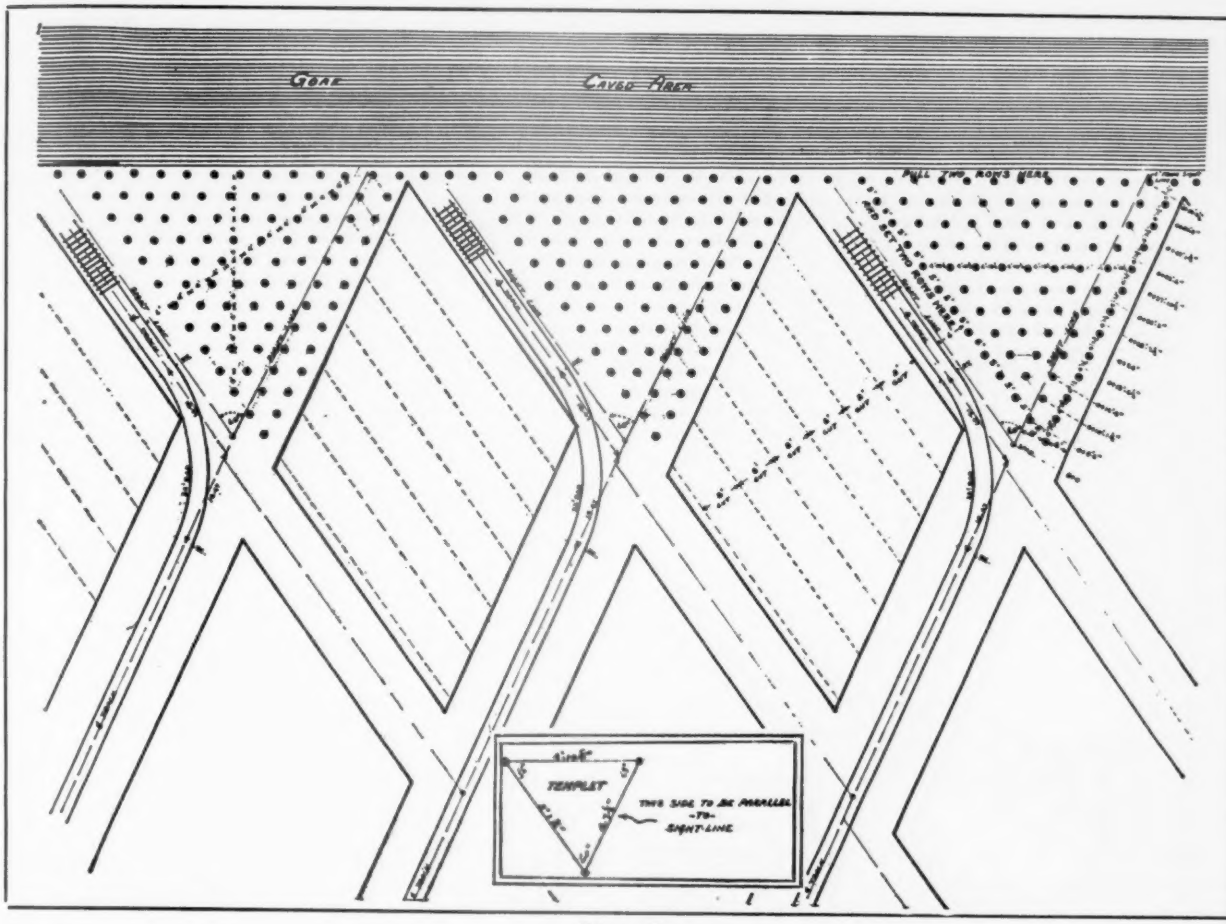


Figure 5
Timber layout—Pillar section

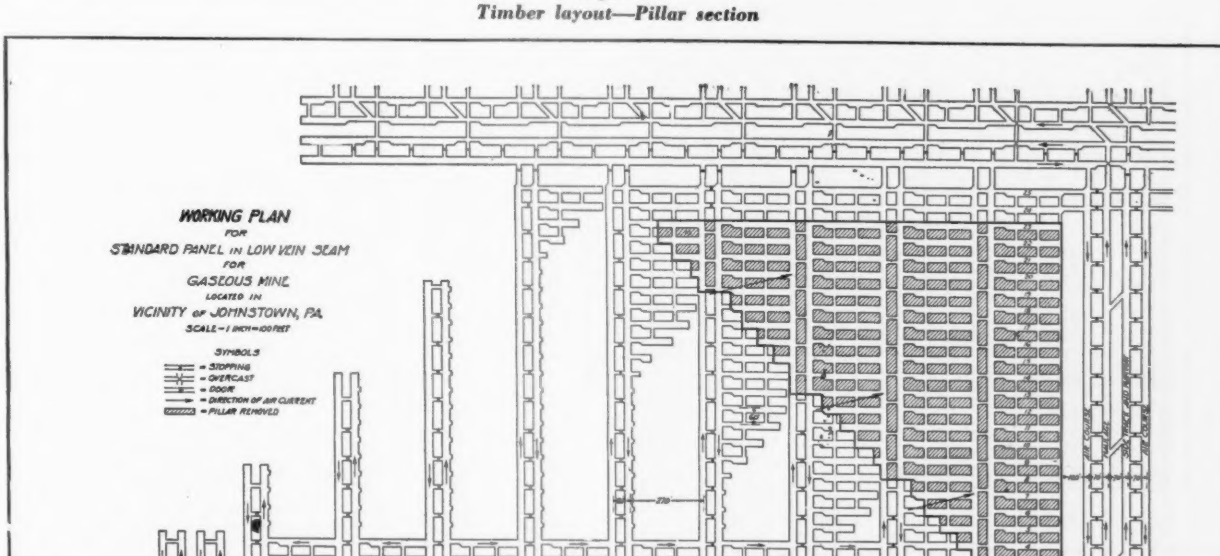


Figure 6

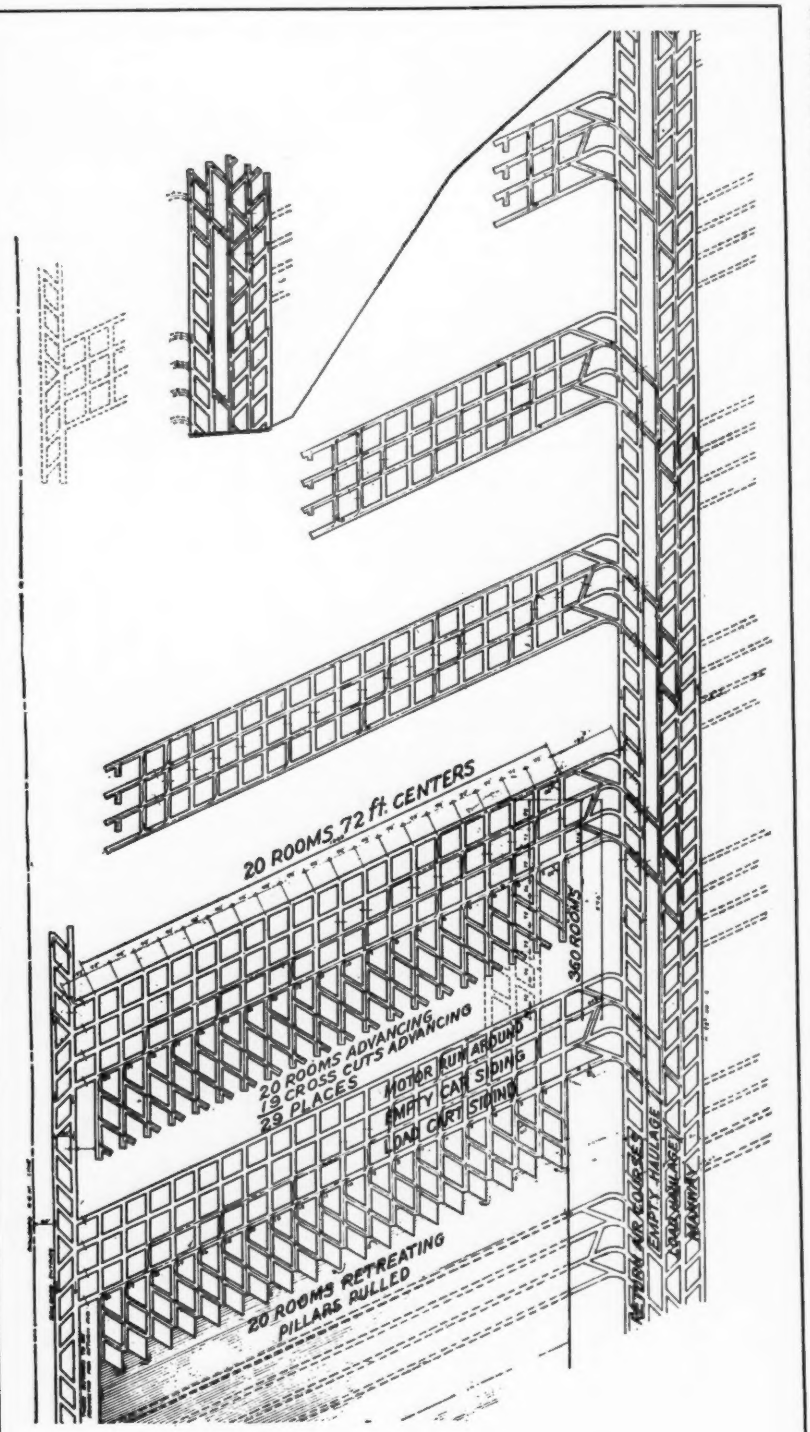


Figure 4

20 room places advancing—16 ft. wide	45 tons—	900 tons	
29 crosscuts—12 ft. wide	35 tons—	665 tons	
19 rooms retreating	120 tons—	2280 tons	
			3845 tons
3 sets room entries, 7 places each	245 tons—	735 tons	
1 main face entry, 8 places average	280 tons—	280 tons	
			1015 tons
Southwest territory		1000 tons	
Main east (one shift)		350 tons	
			1350 tons
Total			6210 tons

(Gross tonnage exclusive of double shifting and second north set of entries)

machine, all places being prepared in this manner by a crew at night. Each machine on rib sections has a crew of 24 men—2 foremen, 5 timbermen, 4 cutters, 2 drillers, 4 trackmen, 2 motormen, 2 snappers, 2 operators, and 1 shotfirer. Under fair conditions the machine will load an average of 500 tons per shift, and the record run to date is 825 tons (165 cars) for an eight-hour shift.

The smaller type machine has an operating crew of 12 men, 1 foreman, 2 cutters, 2 drillers, 2 trackmen, 1 motorman, 1 snapper, 1 shotfirer, and 2 operators.

The fourth method of mining is illustrated in *Figure 6*, a standard working plan for a thin bed, the mine being worked on the panel system, each panel containing nine room headings of 25 rooms each. Room headings are turned on 270-ft. centers, and all places (rooms, entries, and air courses) driven on the advance are 25 ft. wide. Rooms are necked on 60-ft. centers. A barrier pillar of 100 ft. is provided on each side of the panel. This makes a working panel of approximately 3,300 ft. by 1,700 ft., and determines the location of the mains and crossmains.

Each panel is worked on a modified full retreat; that is, no rooms are turned except Nos. 3 and 4 rooms until the room entry is driven its distance. The two rooms driven are utilized for ventilation. Recovery operations are started as soon as the first room entry is driven its distance by starting five rooms, successive rooms being started as needed to maintain the proper sequence of work. This recovery work is carried across each succeeding room entry as the fracture line advances until the panel is on full production. No effort is made to reach the back end of the panels before recovery work is started, as ventilation pressures can not be so successfully maintained, and it also requires a larger amount of development.

Each panel is ventilated by two splits of air, one over the pillar workings and the other in the development, the point of split being advanced as the pillar line advances. The seam averages 40 in. thick and as it is gaseous, generating methane at the rate of about 500,000 cu. ft. in 24 hours, provision is made for locking the air against short circuiting at each room entry and across the face of driving mains. For this purpose a place parallel to each room heading is driven as far as No. 3 room. A door across the heading outbye this room and operating in conjunction with the door in the chute permits the operating of either one without disturbing the proper coursing of the ventilating current, and provides the air lock at the earliest possible stage of the entry development. Nos. 1 and 2 rooms are not driven until the pillars have been drawn, and are kept open to supplement the returns and to provide a current of air along the worked-out pillars. Nos. 24 and 25 rooms are also kept open for the same purpose and to provide drains for the air that passes over the tops of the pillar falls. In this manner the air that ventilates the inaccessible and therefore the unexamined portion of the falls is bled directly into the main returns and away from all live workings. No trouble has yet been experienced in passing air over any of the falls.

Rooms are driven 25 ft. wide, the track

(*Continued on page* 18)

MINING SYSTEMS in Indiana and Illinois Adapted to Mechanical Loading

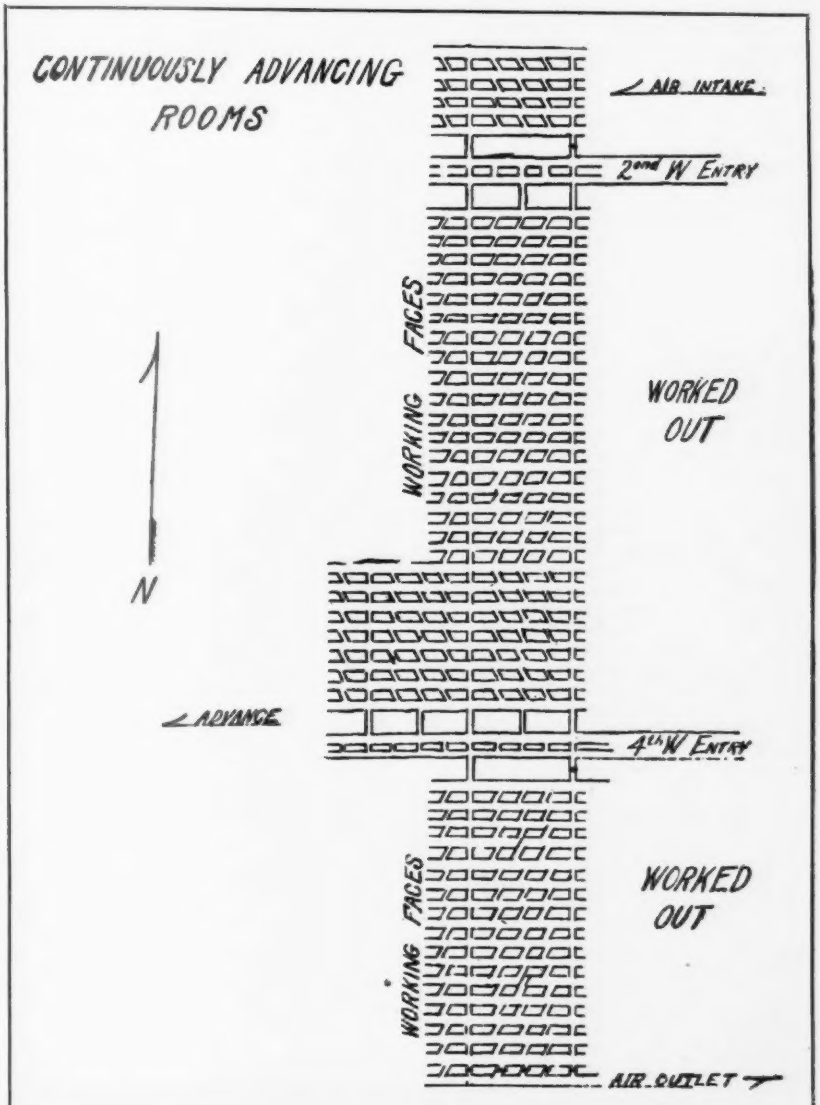
By I. D. Marsh*

MINING systems in Indiana and Illinois as a subject for presentation and discussion takes in entirely too much territory. It was not intended, as I take it, that review of all the many systems used in these two large coal-producing states be attempted. Probably nearly every operation today is working out some changes, great or small, in their methods of mining. The progress made is a real tribute to the organizations at work in these mines.

Mechanical loading has forced everyone to consider and experiment with new mining methods. One of the most successful operations in this district is now, after two years of mechanical loading, changing their entire mining methods. Because I have been for more than three years considering only the problems of a mechanical loading mine, this paper is forced to consider mining methods which apply most peculiarly to mechanization. Rather than take your time explaining in great detail methods which are common knowledge, I am, in the following paragraphs, giving briefly the basic ideas behind two, to my knowledge, new systems. However, I would not be surprised to be informed that several of you here are working along similar lines. I am not making any sales talk for these systems. Your understanding of mining methods will enable you to apply these ideas to your conditions where they may or may not fit.

Before confining myself to specific methods, let me devote to those who might be entirely unfamiliar with Indiana and Illinois a short paragraph upon this topic generally. There are strip or open mines in the north, central, and south parts of this territory producing, roughly, 10 percent of the total production. In the north part of Illinois there is a well-developed longwall operation. There are several scattered operations of retreating longwall, for the most part in small panels, with mechanical loading. The Old Ben Coal Corporation has one of the operations of this type, which I enjoyed visiting. This would furnish a fine paper in itself, but I am not qualified to dwell upon it. There is still a good number of room and pillar operations, but the most popular and widespread method of mining in this territory today is the panel system. You are all familiar with it and its many variations, and there are many beautiful maps being laid out in full size in their carbon medium in Indiana and Illinois which are a credit to their engineers and a joy to their owners.

Probably the most extensive system used today, especially in Illinois, has been developed for mechanical loading. It is a straight panel system with opposite pairs of panels off the entry. An average panel being 12 to 15 rooms deep with 250-ft. length of rooms. One loader develops a pair of panel entries until there is territory in each panel for a loader. It also develops the entries to the next panel at this time. When the next panel is uncovered and there is enough territory behind for a loader in each panel, this loader moves forward to the new panels and continues development. One loader then produces in each



* Alcoa Ore Company.

panel left and develops the panel entries until there is room for two loaders in each panel. At that time a second loader is moved in and that panel is in full production until so many rooms are finished that there is not territory left for both when one loader is left to finish up. The largest problem is to keep the speed of development just right to provide territory as needed. Length of rooms and length of panels must be balanced against length of undercut, speed of loading, and local conditions.

One mine has two unusual developments which I will describe. The general conditions of this mine are as follows:

Cover, ft.................... 80 to 120
Gas .... None
Height of coal, in...66 to 76 and 71 avg.

Top: Varies greatly; 50 percent slate, which is good to very poor; rock from very good to very bad.

Bottom: Extremely soft clay generally, though some rock bottom is found.

Water: About 20 percent of the faces wet and must be loaded out daily to be able to handle with caterpillar-equipped loaders.

Operation (percent): Mechanical loaders, 65; conveyors, 35.

Grades: Generally level, but subject to short local grades up to 6 percent slope.

The first development primarily for high extraction has given some interesting results. Conveyor loading eliminated yardage and all bonuses for narrow work and made possible the development of 80 acres by a two-entry system cutting the coal into solid blocks 100 by 200 ft., with narrow entries on all sides. This development was nearly as low in cost as other conveyor territories at work at the same time.

Starting at the boundary of the mine, the blocks were slabbed 180 ft. long, leaving a 20-ft. pillar on one end. Also small pillars or stumps of coal about 10 ft. square were left across the face where needed for safety, and later shot out. This work over a long period was the cheapest conveyor coal obtained. Extraction was over 80 percent of the coal. In this work the track was laid along the 200-ft. face and connected up at both ends. The cutting machine cut 180-ft. face, which was loaded out. On the second cut the cutter pulled out and resumped, cutting five places 28 ft. wide and leaving four sumps 10 ft. wide. The third cut was as the second, except the cutter would grip in behind the stumps. The fourth cut could usually be straight across the entire face, and after this was loaded out, the track, which was on steel ties, would be relayed behind the stumps. As no switches were used, this was not a heavy burden on track laying. The conveyors could reach the track from the face at all times. This would be repeated until the block was finished. The roof rock had to be drilled and shot to form a break behind the operation.

The second development is "Continuously Advancing Rooms Parallel to the Entry." Three years ago mechanical loading replaced hand loading, and working territories were reduced about 50 percent. Eighteen months ago it was found that the majority of territory developed and left idle would cost an unwarranted sum to clean up and start producing coal. One of the major troubles was that heavy slate, which would not come down when first exposed, could not be held up after several months because of soft clay under the crossbar legs. Entries gave no room for slate disposal, and it would have to be loaded out.

These conditions necessitated research for a better system. Two pairs of entries had been developed and were used as the start of this development. They were parallel and 1,800 ft. apart. Each pair were protected by a 100-ft. barrier pillar on each side. A cross entry was driven between them, which also extended south 1,600 ft. beyond the fourth west. This cross entry also was driven north beyond the second west 280 ft. Rooms were driven west on 60-ft. centers from this cross entry parallel to and on the same face as the main west entries.

The layout is then as follows. Starting at the north end of the entire territory, we have the following:

First: Four rooms.

Second: The second west entry and its back entry protected with a 100-ft. barrier pillar on each side.

Third: Twenty-five rooms between the second west and fourth west.

Fourth: The fourth west entry and its back entry protected with a 100-ft. barrier pillar on each side.

Fifth: Fifteen rooms. The room on the extreme south has solid coal to the south of it and is driven narrow for future haulage.

All rooms are on 60-ft. centers with rooms 25 to 28 ft. wide. Crosscuts between rooms are on 80-ft. centers with 60-ft. length of pillar between crosscuts. Sights are carried in all rooms and entries, and the direction is west.

All room faces are carried equal on a north and south line, with the following exception: The fourth west entry, its back entry and the eight rooms immediately north of these entries are developed 320 ft. ahead of the rest and kept at that distance. This is an aid to haulage and makes it possible to keep the main parting close.

Haulage

Both the second and fourth west entries are main haulage roads and have large partings immediately behind the cross entry. All coal has been coming off the fourth west parting, but local grades have made this an uphill gathering pull, and the second west parting is now taking some of the coal.

The cross entry is the inside haulage road and room switches are turned off the cross entry. Haulage is relayed by locomotive with all conveyors and loading machines served by mules.

Every fourth crosscut or every 320 ft. the crosscuts are lined up on sights and become the new cross entry as the faces move beyond it. All track is removed behind the new cross entry as early as possible. The main entries are mined with the rooms on one side and the barrier is cut on that side every 160 ft. The barrier on the other side of the main entries is cut only at each cross entry, or every 320 ft. A fresh-air course leads directly to the four rooms in the extreme north, and the air sweeps south over all faces, being restricted by stoppings in one of each pair of barrier pillars, and also in the crosscuts to the last room in the south.

Operation

Mechanical loaders operate in those territories most suited to them with conveyors keeping up the heavy slate, extremely wet and bad rock conditions. Loaders are frequently changed from one set of rooms to another as conditions change.

The continuously advancing room system does not need any development work away from the main producing territory. It offers a battery of working faces which can be supervised from end to end without leaving the producing faces. All operations such as cutting, drilling, track, deadwork, and timbering can be quickly shifted from one unit of production to another without lost time. The problem of starting and finishing new panels or territories is completely eliminated. It does not give protection against conditions where a squeeze would ride up to the face. A squeeze caused by excessive extraction gave some concern in this operation, but the work traveled so fast that it caused little inconvenience and did not progress when rooms were made their proper width again. These rooms are now approximately 1,500 ft. deep, and the program is to continue them about 2,000 ft. further to the boundary of the coal. This layout, though not entirely developed as planned at this date, has eliminated the delays in mechanical operation caused by territories working out and new territories opening up.

Pennsylvania Mining Systems

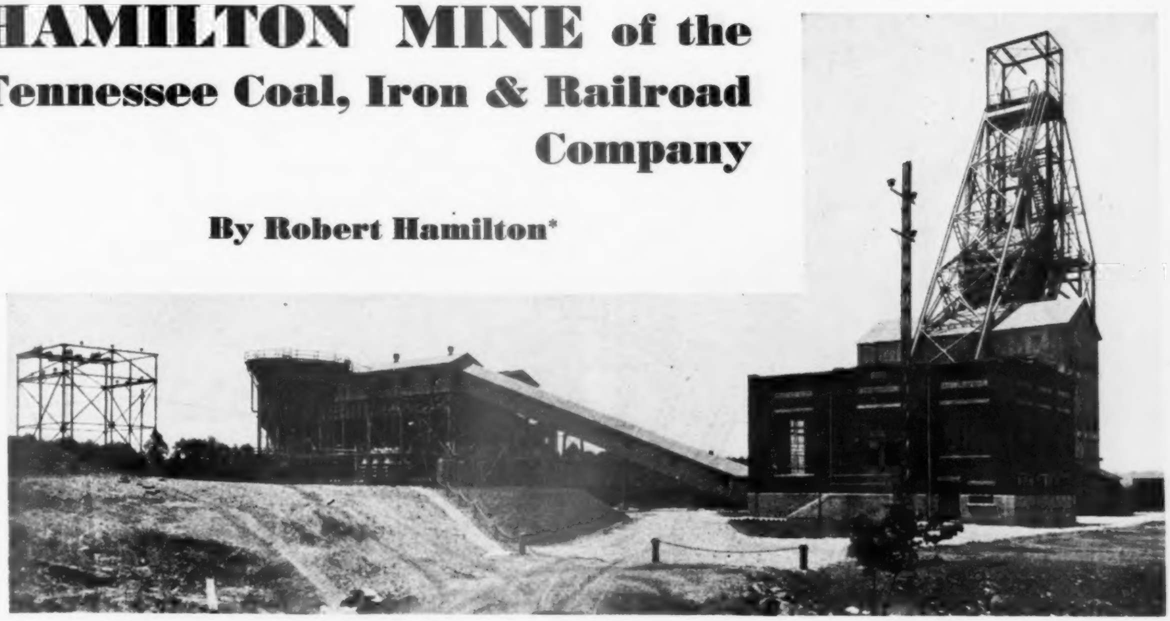
(*Continued from page* 16)

being laid 4 ft. from the rib that is to be drawn. The roof over the gob side is supported by three props set on 4-ft. centers in rows 6 ft. apart. A fourth prop is set in this row not over 1 foot from the rib. The props are set 1 ft. from the face before the coal is shot so that the greatest distance between the props and the face at any time is 7 ft. A safety prop is also carried ahead of the car while the cut is being loaded. Air courses and headings are timbered in a similar manner, except that in the case of the headings the road props are doubled up. Cut-overs in pillars are cut 12 ft. wide, leaving a stump of 8 ft. that is extracted by pick work. Timbering in these pick places is set on 4-ft. centers.

Each panel is designed to produce about 500 tons of coal and employ about 70 miners. Gathering is done by permissible type storage-battery locomotives, as bare power lines and open-type locomotives are not permitted in the split air current. These motors deliver empties to the headings and gather the loads to the heading chutes, where they are picked up by a shuttle trolley locomotive and taken to the side track, where it is met by the main-line locomotive.

HAMILTON MINE of the Tennessee Coal, Iron & Railroad Company

By Robert Hamilton*



The Hamilton Mine plant from east side, showing hoist house and tipple

THIS mine is located close to the southeast boundary fault of the Warrior coal field, approximately 6 miles from the center of Birmingham.

The total area of workable coal is approximately 3,920 acres, carrying about 30,000,000 tons of recoverable coal. This area was determined by diamond drilling. Seventeen holes were drilled to an average depth of 620 ft. Proximate analysis and distillation tests were made of coal cores recovered. A number of dislocation faults intersect the area, but as the Pratt seam (540 ft. above the Mary Lee seam) has been worked out, the location and throw of each fault has already been proven. The coal seam contains from three to six slate partings, the thickness of the coal and the slate partings being very variable.

	Data from drill hole records	From mine workings
Average height of seam........	7'-0"	
Average thickness of clean coal	5'-8"	5'-5"
Average thickness of slate partings	1'-4"	1'-7"

During 1920 the coal requirements exceeded the capacity of existing mines and it became necessary to develop the Mary Lee seam as quickly as possible, by means of a slope opening from the outcrop of seam. As the location was not suitable for successful operation during probable life of mine, a coal washery was not erected, and the greater part of the equipment secured from abandoned mines. However, as the proposed final location of hoisting shaft, coal washer, etc., was already determined, the mine entries, etc., were arranged accordingly.

During 1927 work was started at the shaft location, but only the essentials for operation were completed; i. e., the railroad side track, hoisting shaft, head frame, main hoist, and coal washer; the office, employes' bathhouse, and other buildings at slope opening still remaining in service.

The work entailed was completed during 1928 and hoisting operations started in November of that year.

The hoisting shaft is oval-shaped and is concrete lined throughout, the maximum dimensions 21 ft. by 12 ft. in the clear, with a total depth of 644 ft. to bottom of sump.

Detailed drawing B-290 shows the arrangement for transfer of coal from mine car to hoisting skip.

The main hoist is of the single-drum type, gear connected to a 600-hp., 2,200-volt, A. C. motor. To reduce the starting torque, the drum is coned on both ends, increasing from 7 ft. to 10 ft. in diameter in the first four rope turns. The drum is grooved for 1½-in. rope, rope travel 700 ft., maximum rope speed 1,070 ft. per minute. Estimated capacity, 360 tons per hour. The hoist is equipped with Lilly control and devices for prevention of overwinding.

The plant is designed for an average daily output of 5,000 tons when operating on double turn of eight hours each. On November 4, 1930, the night turn hoisted 2,943 tons.

Method of Working

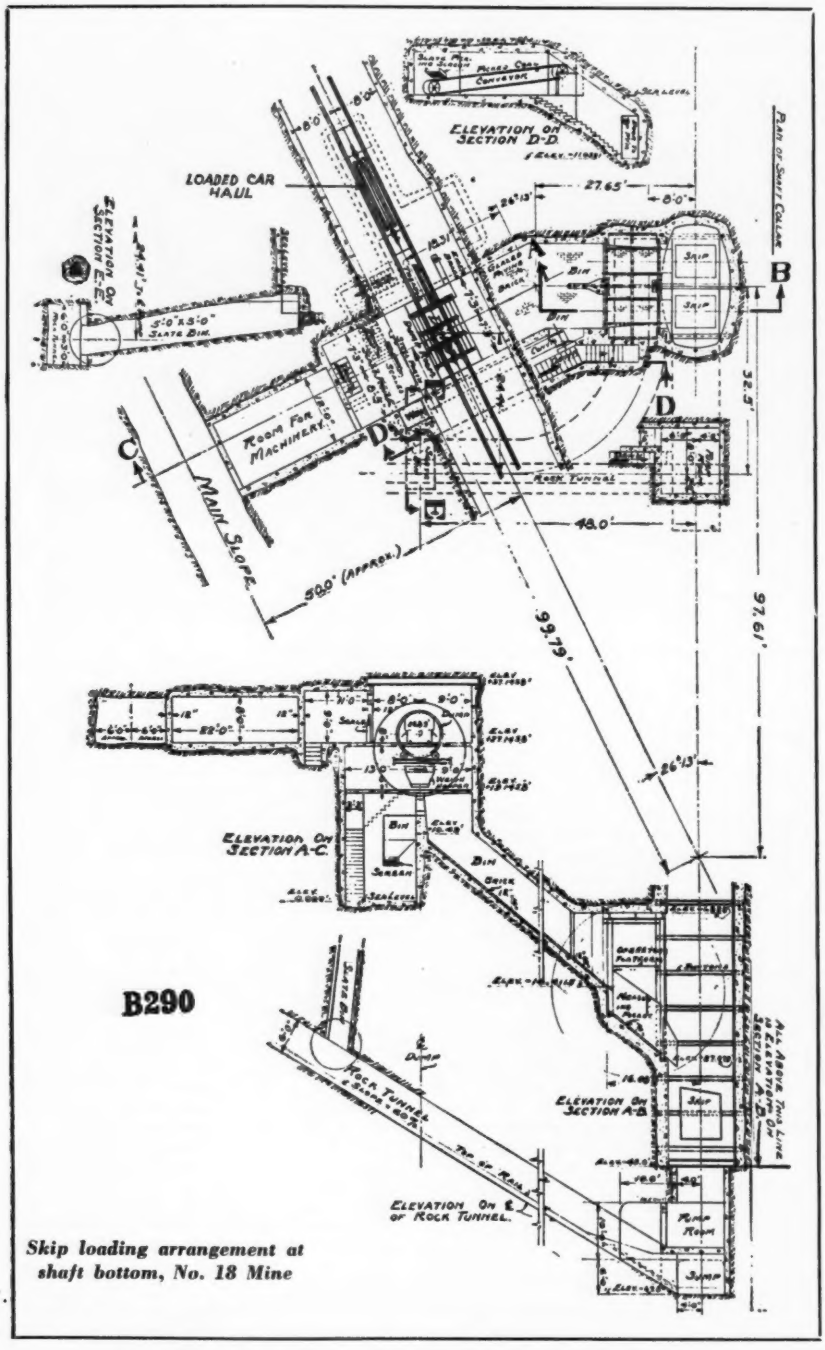
The mine was originally opened up in 1920 with a slope, manway, and two air courses driven from the outcrop. Room entries were turned to the right and left of the slope to work out the pitching territory. When the basin was reached panel headings were turned about 1,600 ft. apart with double room entries driven both ways connecting the panel headings. The rooms are turned on 82-ft. centers and driven 28 ft. wide to a depth of 300 ft., leaving a 54-ft. pillar between rooms. This method of working extracts 46 percent of the coal on the first mining.

All the coal is undercut by machine, blasted down by permissible explosives, and loaded by hand.

The rock parting is picked out and gobbed in the center of the room between the two tracks which are laid close to

The coal washing plant at the Hamilton Mine

* Consulting Engineer, Tennessee Coal, Iron & Railroad Company.



B290

Skip loading arrangement at shaft bottom, No. 18 Mine

the rib on both sides. Entries and air courses are driven 20 ft. wide with a 30-ft. pillar, which allows the rock to be gobbed on one side of the track. Side pillars are left on each side of the panel headings from 100 ft. to 125 ft. thick. The undercutting is done with short-wall mining machines using 7-ft. cutter bars. Water pipes are installed in the cutter bars so that the water sprays out at the front end of the bars and wets the cuttings. In the room entries and rooms the track centers are carried 5 ft. from the rib. The timbers are set on 4-ft. centers both ways and are kept 30 in. from the rail and 4 ft. back from the face with a safety timber set in the front of the roadway.

Haulage

Hamilton mine territory is traversed by five faults, which make it necessary to work the coal between any two faults to suit the general pitch of the seam in that area. The faults also cause considerable grading work to be done and rock tunnels driven for motor haulages to the different levels. On account of the rolling nature of the seam and the weight of the cars, it is necessary to pull the cars in or out of the rooms, and mules are used for this purpose. Small hoists are used to pull the cars out of the entries that are going to the dip. All other hauling is done with electric locomotives. Each panel heading is provided with a side track to which the gathering locomotives deliver the cars and the main-line locomotives haul from the side tracks to the dump at the shaft bottom.

There are in use one 4-ton, one 6-ton, one 13-ton, ten 8-ton, and one 15-ton locomotives, or 14 in all. The mine cars are built of wood with a steel frame bottom. There are in use 217 old type, 1½-ton, and 350 new type, 2-ton, cars.

The tracks in the main slope and panel headings are laid with 60-pound rail on tie plates and creosoted ties spaced 24-in. centers. All 60-pound tracks are laid to grades furnished by the engineering department. The tracks in the room entries are laid with 40-pound rail on oak ties, and the rooms are double tracked with 16-pound rail on small pine or oak ties.

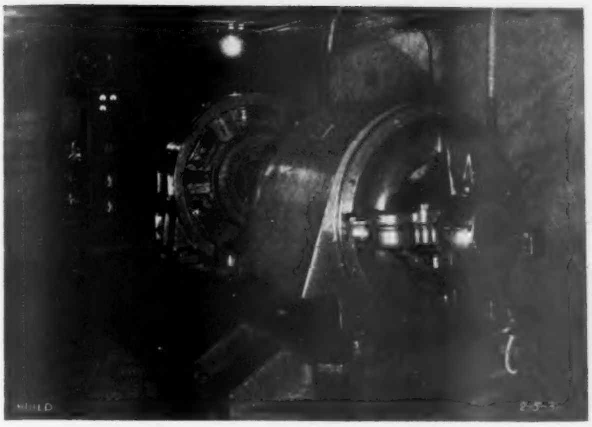
Shaft and Dump

In 1927 a 12 ft. by 21 ft. by 644 ft. oval concrete-lined shaft with steel buntons and wood guides was sunk 5,200 ft. from the pit mouth. The shaft is equipped with two steel self-dumping skips of 6 tons capacity each.

When hoisting was started at the shaft, the slope was continued in use as a material-way. The manway driven parallel to the slope is still in use as a travel-way, for men entering or leaving the mine. It is proposed in the near future to sink a rock slope for a material-way and manway near the shaft.

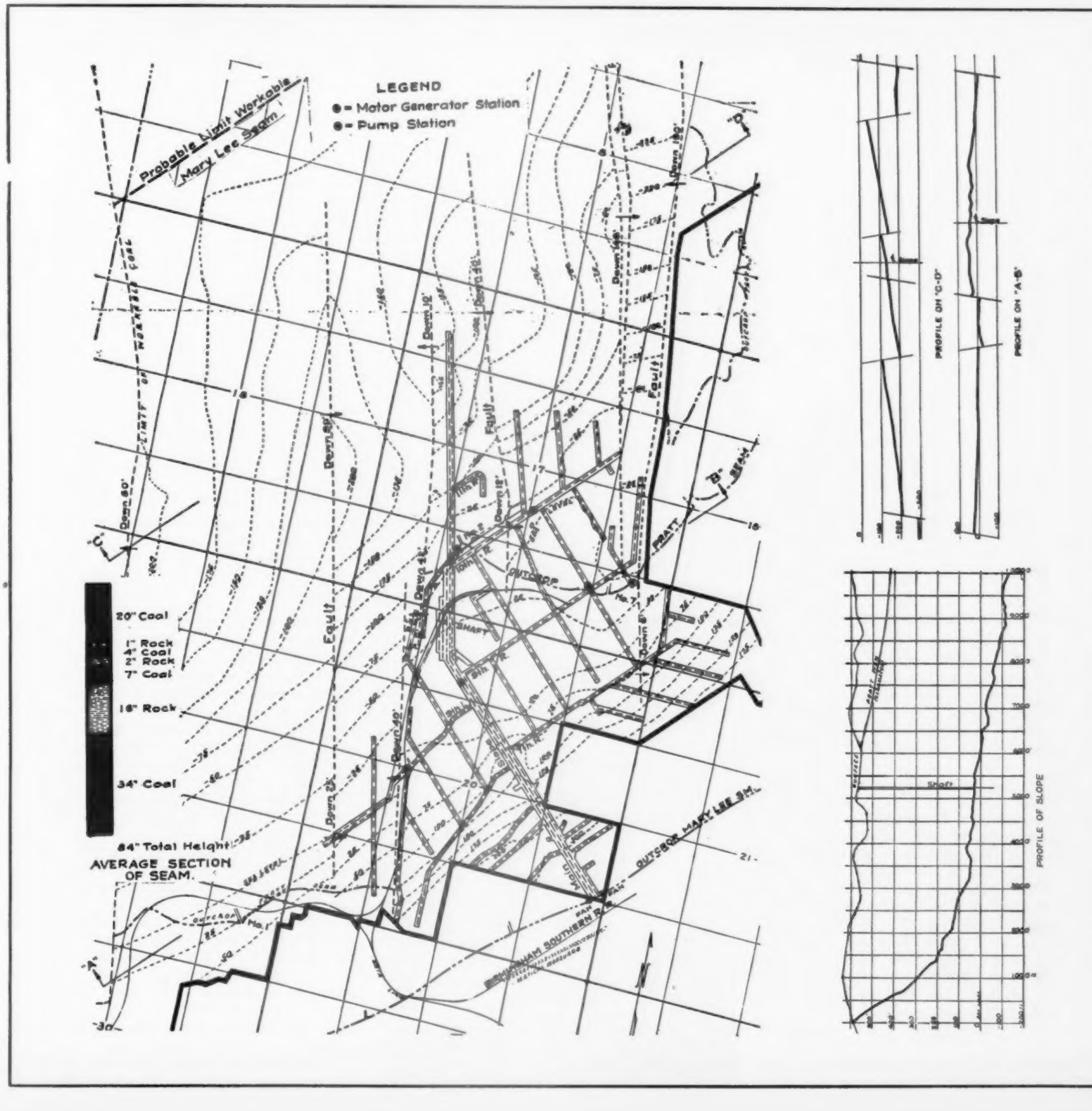
The dump, located at the shaft bottom, is of the single-car revolving type, motor driven, with a capacity of six cars per minute. It is equipped with a car stop which catches two cars on the loaded end, one in the dump and the first empty car. Swivel couplings are used so that the cars are not uncoupled while being dumped. A short car haul at the loaded end feeds the cars into the dump. The operator's stand is also at the loaded end of the dump, allowing him to operate both the car haul and the dump. Directly below the dump is the scale hopper which empties into the skip loading bin or into the slate-picking bin, as is desired. The doors on the bottom of the scale hopper are operated by air cylinders, with the control valve located on the weighman's desk in the scale room. This room is completely enclosed and contains the scale beam, with a Streeter-Amet recording attachment.

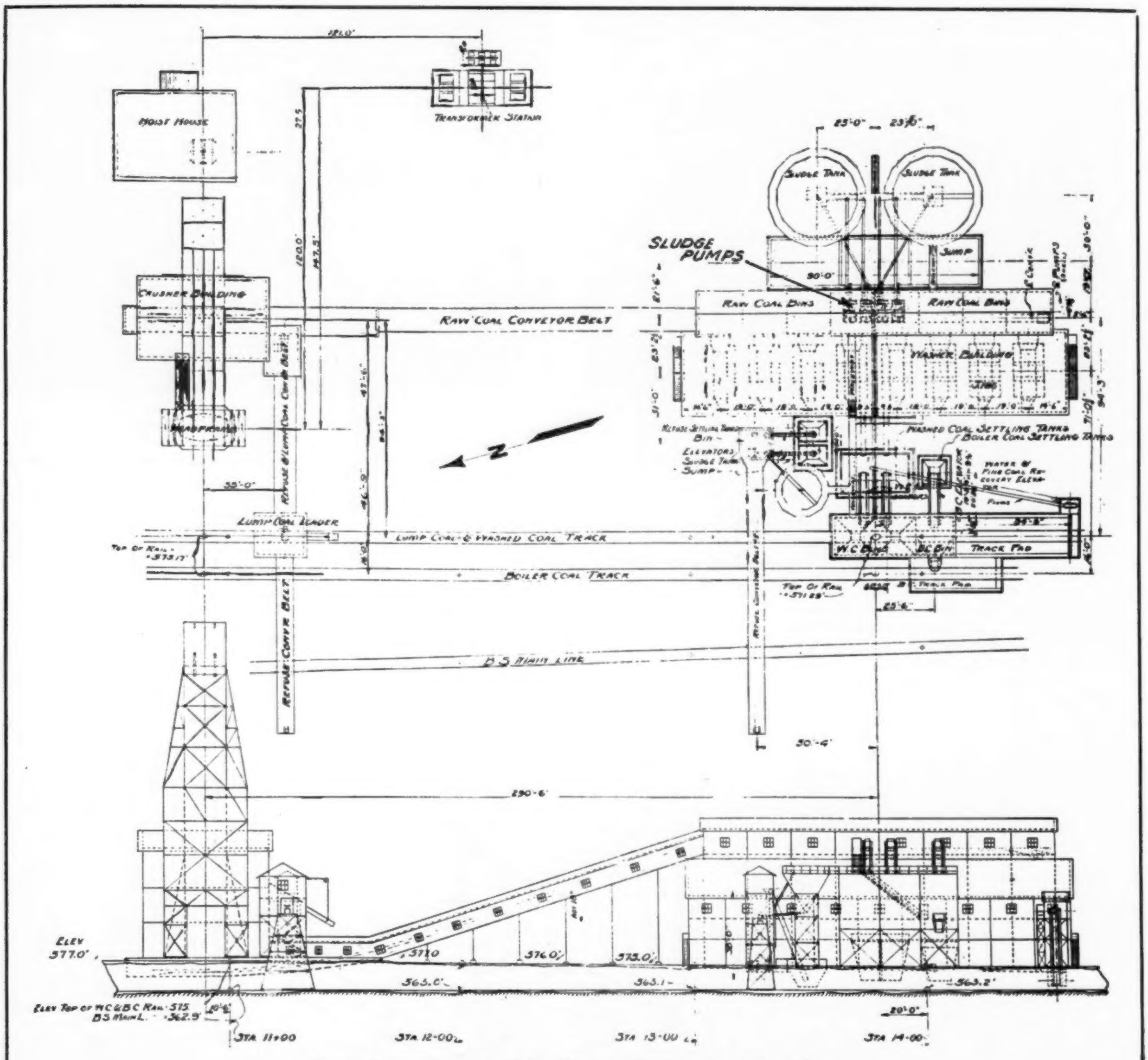
The skip loading bin empties into two measuring pockets of 6 tons capacity each. The pockets are filled and emptied by a door on the front and back of each, operated by hydraulic cylinders. The skip loading operator's room is located at the side of the shaft, just above the measuring pockets. When the skip comes to the bottom, the operator opens a four-way valve in the water line, causing one cylinder to close the back gate and shut off the coal from the main bin. As soon as this gate is closed the other cylinder opens the front gate, allowing the coal in the measuring pockets to pour into the skip. Turning the four-way valve back to its original position allows the pocket to fill with coal again. One valve is located on each side of the operator, while in front is a panel with lights showing when the skips are at the bottom ready for loading, when the front gates are closed, when the measuring pockets are full, and when the bin on the surface, into which the skip dumps, are full. A telephone, speaking tube, and signal to



Portable type motor generator set, Hamilton Mine

Fixed type motor generator set





General arrangement of headframe, tipple and coal washery

the hoisting engineer are also on this panel. In the slate-picking room equipment is installed to handle about 55 cars per shift over a picking screen. The slate is picked out of the coal and measured, and the loader of each car is docked according to the quantity of slate obtained.

At the bottom of the shaft is a sump and pump room with a small triplex pump, automatic in operation, to keep the water pumped out. Above the sump the shaft is floored with a steel floor set on a slant to form the bottom of a spillage bin for catching all the coal spilled into the shaft. The bin is emptied into cars through two chutes. A rock tunnel, driven on a 60 percent grade, connects the bottom of the shaft with the coal seam. This is provided with a track and a small hoist at the top to handle the cars to the spillage pocket.

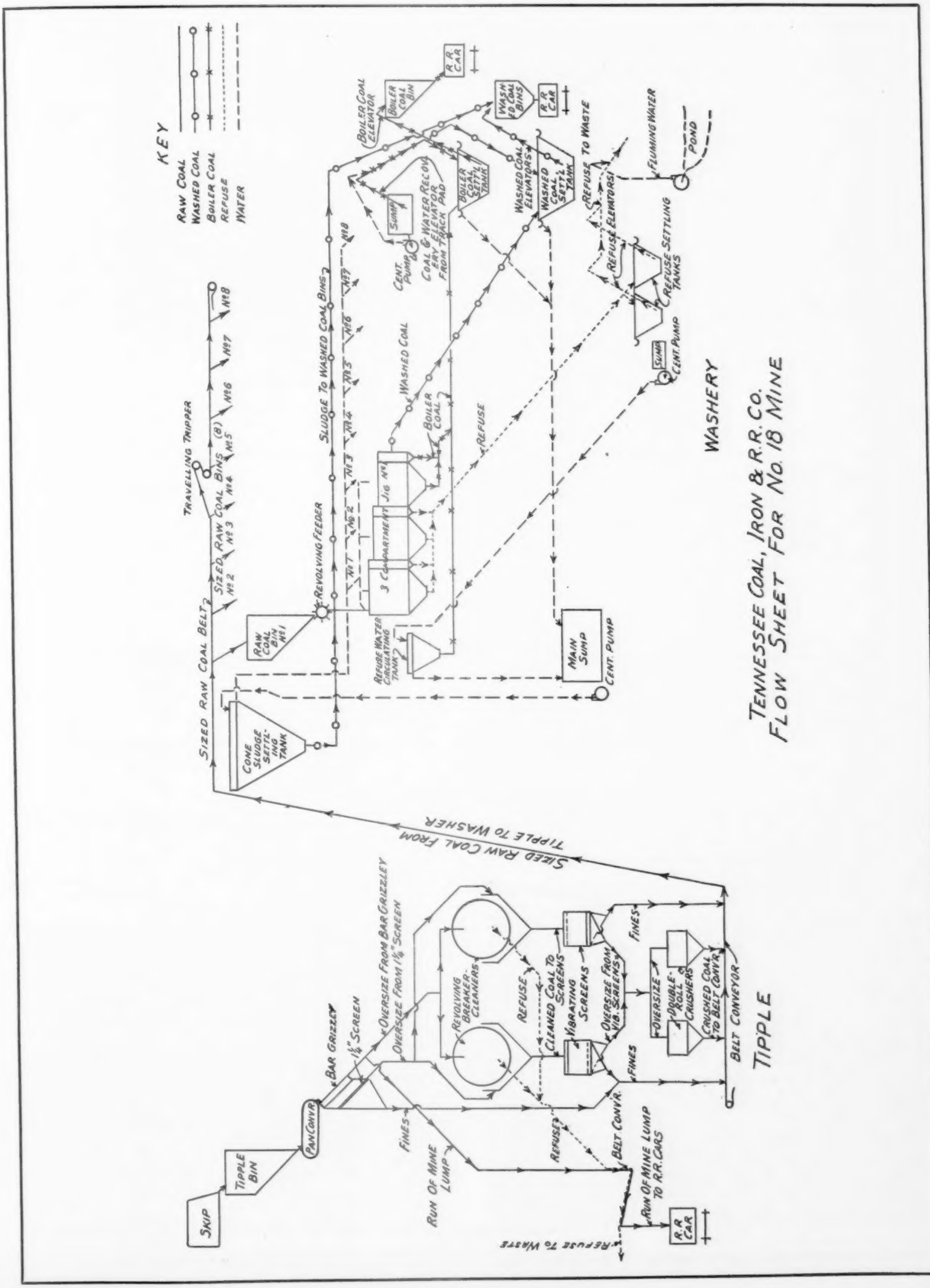
Ventilation

The slope, manway, and shaft are used as intakes, and the two air courses, one on each side of the slope, are used for the return air.

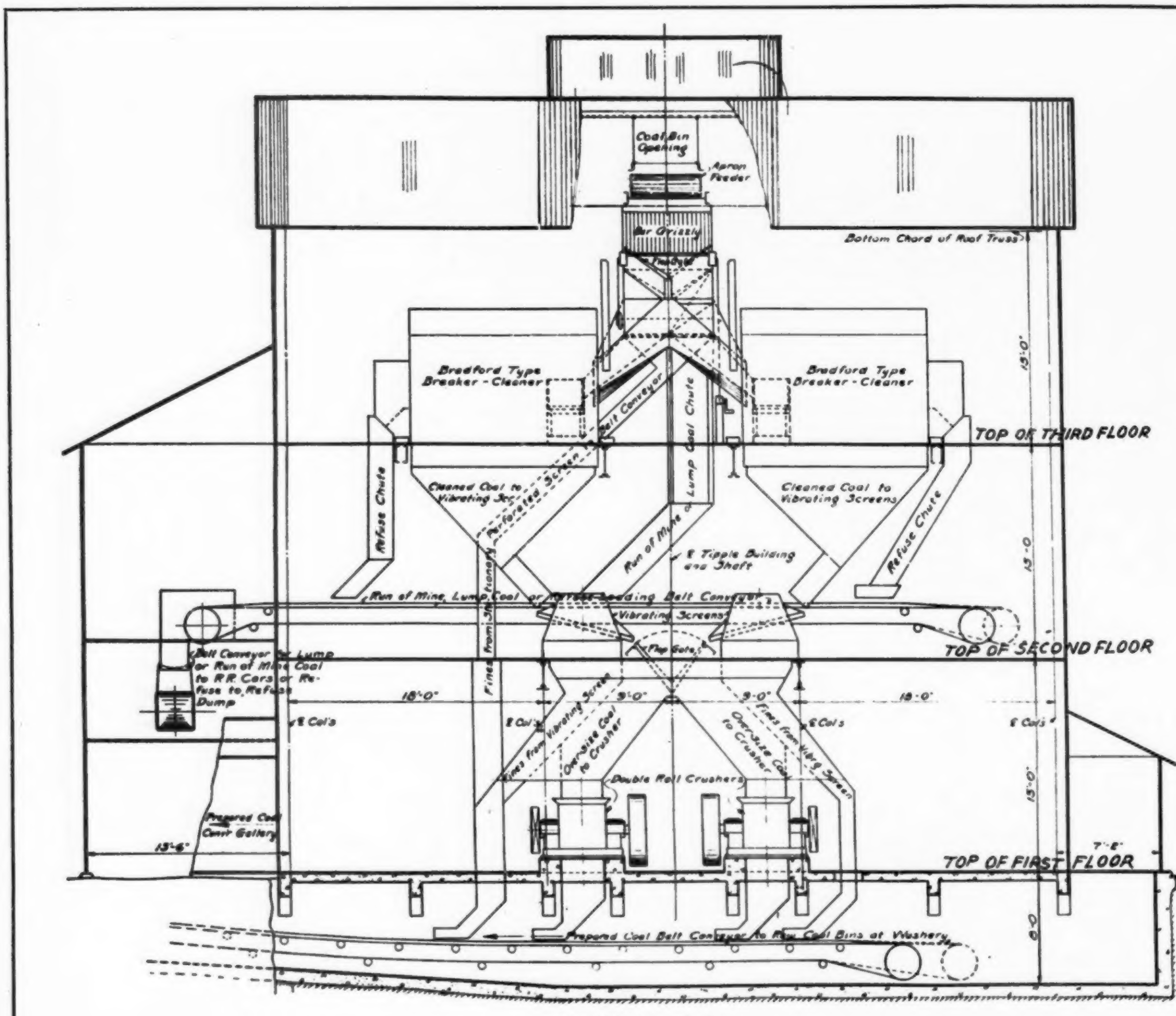
At the pit mouth are located two exhaust fans, one connected to each return. Both fans are belt driven and the drives are similar. On one side a 125-hp., 2,300-volt motor for regular operation, and on the other side a steam engine for an auxiliary drive. On each end of the fan shaft friction clutch pulleys are mounted, so that either drive can be put in service immediately. The left side of the mine is divided into three splits, and is ventilated by one fan, while the right side is divided into four splits, and ventilated by the other fan.

The stoppings are built of rock and plastered on both sides with a clay and cement mortar. Doors are built in pairs. One door being fastened open and used only as an emergency door. The overcasts are made with a large area and are built of concrete. The air is carried

Head frame with screening and crushing building, Hamilton Mine



Tennessee Coal, Iron & R.R. Co.
Flow Sheet For No. 18 Mine



Longitudinal Section.

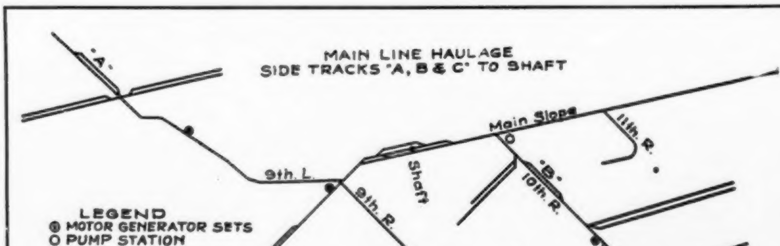
Arrangement of equipment in steel crusher building

from the last cross-cut to the face of each entry by line curtains, and is forced into the rooms by block curtains on the entry. The mine air is sampled once each week with a Methane detector in a large number of places in the workings, in the return from each split, and in each fan. The volume of air is measured at the same time and a report is made up showing the places taken, volume of air, percent of methane, and the cubic feet of methane per minute.

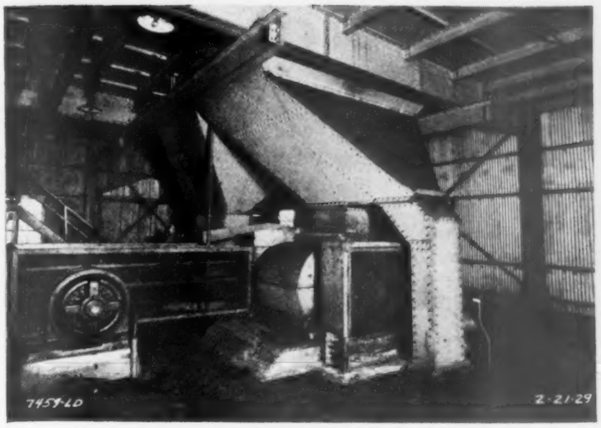
Once each month the amount of air is measured in the last cross-cut in the face of each entry, the intake on each split, the total intake for the mine, and the return at each fan. From this data a report is made showing the place taken, the volume of air, temperature, humidity, and the number of men working in each entry.

Power System—Surface

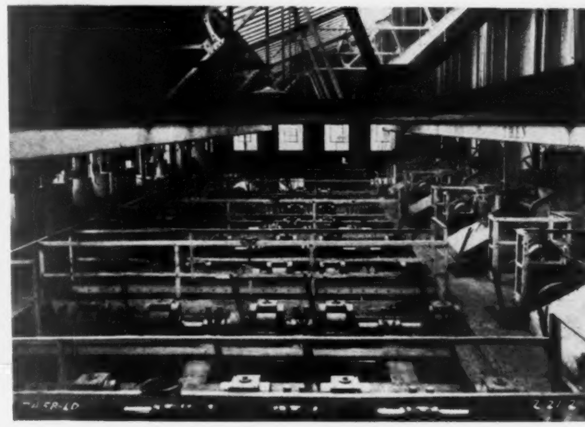
Power for Hamilton mine is obtained from the central power plant at Fairfield Steel Works, through a transmission line built with steel poles and carrying 44,000-volt, 60-cycle power to a transformer station near the shaft. In this station are three 400-k.v.a., 44,000/2,300-volt transformers for mine power feeding through cables down the shaft. There



HAULAGE DIAGRAM
TENNESSEE COAL, IRON & R.R. CO'S
HAMILTON MINE
BIRMINGHAM, ALA.



Crusher floor, Hamilton Mine washer



Jig operating floor, Hamilton Mine washer

are also three 200-k.v.a., 44,000/220-volt transformers for surface power in the tipple and washer. The 44,000-volt line extends from the shaft to the pit mouth, where there are located three 100-k.v.a., 44,000/2,300-volt transformers furnishing power to the two fan motors.

Power System—Underground

The armored cables extending down the shaft enter a switch room containing oil circuit breakers. From this room one armored cable carries 2,300-volt power to the main pump station, while four other armored cables, going in opposite directions, each carry 2,300-volt power to four motor generator stations. Three of these stations are each equipped with one 150-kw., 250-volt, D.C., full automatic stationary motor generator sets, while the fourth has one motor generator set of the same size but of a portable type.

The return for D.C. power is through the track. On 60-pound track each rail is bonded with two 4/0 concealed bonds. One bond is placed on each side of the rail and passes under the splice bar. On the 40-pound track each rail is bonded with one exposed bond.

Rock Dusting and Spraying

All the main haulage roads are rock dusted on an average of every two weeks. Samples are taken twice a month, and if the inert material in any entry shows less than 65 percent, that entry is immediately redusted.

All the room entries, rooms, and the inside end of panel headings are equipped with spray lines and are washed down daily with water. In all the working places the pipe lines are kept near the face, so that the hose from the mining machines can be attached to the spray line. If for any reason there is no water on the mining machine, it is considered out of order and is not allowed to cut the place. A safety ground wire is attached to the frames of all electrical machines, such as mining machines and electric drills. The other end being attached to the spray water line. None of these machines are allowed to run without being so equipped.

Pumping

All the water in the mine is pumped by gathering pumps into the main sump or to some point where the water will flow into the main sump. The sump is in a heading driven between the shaft and the fault to the west, with the upper end of the sump area being at an elevation of 45 ft. The lower end of the sump heading is stopped up with a concrete dam with two suction pipes leading to the main pumps, which are located below the lower end of the sump heading, the pumproom floor being at an elevation of 3.5 ft. The water is always above the pumps, so that priming of the pumps before starting is eliminated.

In the pumproom there are two 1,000-g. p. m. centrifugal pumps, 580-ft. head, directly connected to 250-hp., 2,300-volt motors. These pumps are controlled automatically by float switches. They discharge through a drill hole to the surface.

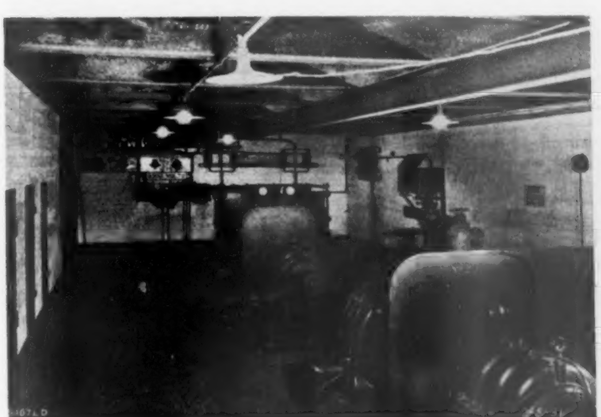
Since the spray water pipe lines for the mine are connected to this discharge pipe, it was necessary to install a small piston pump in the pumproom to keep the desired pressure in the main discharge pipe and spray lines when the centrifugal pumps were not running. This pump is also automatically controlled by a pressure switch, which keeps the pressure within narrow limits at all times.

Hamilton Washer

This plant is designed to prepare 320 tons feed coal per hour, equal to maximum capacity of mine; requiring eight three-cell jigs. While the balance of the plant for this capacity is completed, up to date only five jigs have been installed. To ensure satisfactory operation, the maximum capacity of each jig is limited to 40 tons per hour.

The plant is of steel and concrete construction throughout. All settling tanks are of reinforced concrete, also the main building up to the floor level. All bins, tanks, etc., above floor level are of steel construction, no timber being used.

The jigs are of double-plunger type, with three 6 ft. by 6 ft. cells, equipped with Elmore revolving discharge valves.



Main pump room, Hamilton Mine

Revolving dump. A—Dump control; B—Car haul control; C—Entrance to scale house

Three separations are made; i. e., coking coal, boiler fuel, and refuse. The jigs are on a concrete floor, permitting easy access to all moving parts. Operating platforms of permanent construction are provided at the same elevation as top of jig, where provision is made for regulation of slate or hutch work valves. A central electric control switchboard is also erected on this platform, connected to all motors around the washer.

All the jigs, elevators, crushers, pumps, etc., are provided with individual motor drives, eliminating all shafting and pulleys.

In the ordinary sense of the word, there is no tipple at this plant, as the crusher house is an integral part of the head frame.

The product from the mine is hoisted in 6-ton skips, discharging into the skip bin, the bottom of which is 55 ft. above the feed coal conveyor belt to the washer; this permits a gravity flow of the product from the skip through the preparation plant to the washer feed belt, which delivers the prepared coal to the raw-coal bins at the washer.

A short pan conveyor feeds the product from the skip bin to a bar grizzly with 1¾-in. spaces.

The undersize from this grizzly falls onto a stationary screen plate with 1¼-in. round perforations. The fines from this screen plate passing directly to the washer feed belt. The oversize from the screen plate is delivered to the double-roll crusher.

The oversize from the bar grizzly is delivered to a revolving Bradford breaker. The breaker has 2-in. perforations, through which the broken coal passes to vibrating screens with 1-in. square mesh; the fines from the screens passing directly to the washer feed belt. The oversize is delivered to the double-roll crusher for final crushing, then to the washer feed belt.

It has been found from a recent test that only 1.4 percent of product from the crushing plant remained on a ¾-in. square mesh.

The Bradford breaker rejects large rock, tramp iron, etc., which is delivered to the refuse conveyor. This refuse conveyor also acts as a run-of-mine lump-coal conveyor, which permits the loading of run-of-mine coal. During any emergency this run-of-mine coal is by-passed to the conveyor from the bar grizzly through a trap door in the grizzly chute.

All the preparing equipment in the crusher house is installed in duplicate, with necessary gravity chutes for movement of product to either set.

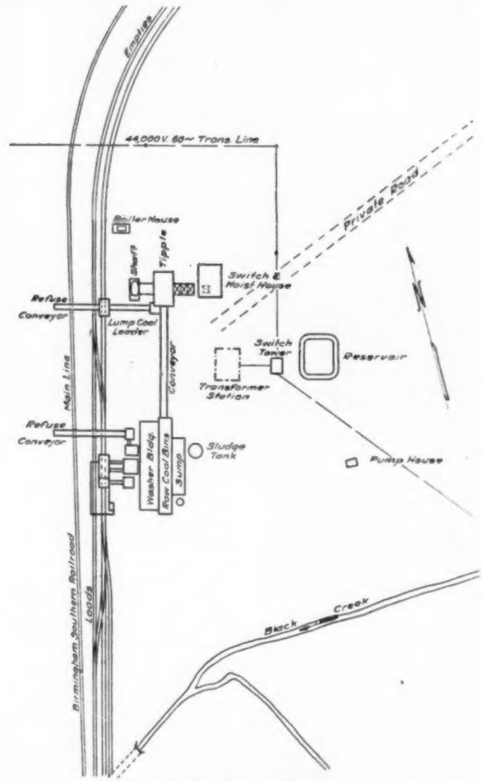
All equipment is carefully housed, and a dust-collecting system installed, which delivers the moistened fine coal dust into the coking-coal bin.

From the crusher house the product is delivered by a 30-in. conveyor belt to the washer feed coal bins, and distributed therein by an automatic tripper.

Adjustable rotary feeders deliver to the jigs. The overflow from the third cell of each jig, carrying clean coking coal, is flumed to a large settling tank, from which it is transferred to the railroad loading bin by a dewatering elevator.

The boiler fuel from the third valve and hutch of each jig is treated in a similar manner.

The refuse from the first and second valves and hutches follows the same routine, but is not loaded in cars, and is disposed of by a combination of conveyor



Surface arrangement

belt and water-flume system. By this system there is practically no danger of the refuse "firing."

Circulating water from the washed coal and boiler fuel settling tanks overflows into the main sump and is pumped therefrom to a large cone-shaped settling tank, known as the sludge tank, for the removal of fine coal. A small auxiliary tank is provided for primary settling of water from the refuse settling tank, to remove the heavier slate particles; the overflow therefrom is flumed to main sump and returned to circulation.

The overflow from the sludge tank is collected into the main pipe that supplies the jigs. By this arrangement the washer water is used over and over again, only the make-up water being added; this is approximately 12 percent of the weight of the feed coal. Approximately 2½ tons of water is in circulation per ton of washed coal.

The "sludge" is drawn off continuously from bottom of the tank and delivered into coking coal loading bin by compressed air, ensuring thorough distribution in finished product.

A concrete drainage pad extends along the railroad from the upper end of bins to a point some distance below the washery. All leakage or drip from either the loaded cars or bins is collected in a small concrete sump, from which it is recovered by a bucket elevator and returned to circulation.

This pad has proven a great advantage in keeping the tracks clean, as all coal spillage can be hose flushed into the sump and elevated into the washed-coal settling tank.

When the washery is in operation, all jigs, settling tanks, etc., are full of water. When it becomes necessary, in case of accident, or the necessity of cleaning out any of the settling tanks, the system is drained.

As pollution of the adjoining creek is forbidden, an extra large sump is provided into which all water can be drained. When the emergency is over the circulating pumps put the water back into circulation.

WASHERY RESULTS FOR NOVEMBER, 1930

Turns in operation 17.		Tons
Feed coal delivered to washer............		20,931
	Tons	
Coking coal—80.66%............	16,883	
Boiler fuel—9.79%..............	2,049	
Refuse—9.55%	1,999	
		20,931

Float in refuse @ 1.40 S.G. = 6.8%
9.55% × 6.8% = 0.65% of feed coal lost in washing.

Analysis

	Vol. M.	Fix. Car.	Ash	Sul.
Feed coal	25.31	58.04	16.65	0.88
Coking coal	27.35	63.46	9.19	0.76
Boiler fuel	25.53	58.33	16.14	0.94
Refuse in float @	1.40—	6.8%	8.01	0.78
Refuse in sink @	1.40—	93.2%	64.16	1.49

Sink and Float Tests—Feed Coal

Separation @ 1.40 S.G.....	78.75%	7.92% ash
1.40-1.56 S.G.....	8.17%	22.92% ash
1.56 S.G.....	13.98%	61.83% ash

Efficiency for Month

7.92% ash ÷ 9.19% ash	= 86.18%	separation
100-0.65	= 99.35%	recovery
Total efficiency	= 85.62%	

Mining Systems in Utah

By George A. Schultz*

SYSTEM, according to the dictionary, is a comprehensive plan.

The writer does not know of any coal-mining operations in Utah that have from the beginning followed any one system. Rather, each operation is worked on several different systems, which have either been copied or devised to meet the different conditions encountered.

While the coal fields of the state are generally favored with good conditions, in so far as thickness of seams, roof conditions, and grades are concerned, these fields present their difficulties to systematic mining.

Among these difficulties are the usual in faults, "wants" or barren areas and sudden changes in grades, and the unusual in tremendous overburden, very thick seams, and perhaps the most highly explosive coal dust in the United States.

The extraction of high coal alone presents numerous problems, but when the high coal is overlaid with from 1,000 to 3,000 ft. of sandstones in beds 20 to 500 ft. in thickness, these problems are considerably increased.

The coal dust presents a greater problem in the high coal, for the reason that the finest and therefore the most explosive dust lodges at the high points, where it is an invisible but constant menace unless constantly and thoroughly kept inert through the use of large quantities of rock dust.

Many of the operations in the thick seams of the state have adapted mechanical loading in some form, for from 50 to 75 percent of their output, but it is doubtful if any of them will ever work on a 100 percent mechanical loading basis.

Many attempts have been made to extract pillars with loading machines, but these efforts have resulted in so many injuries and in such incomplete recovery that most operations now use mechanical loading on advance work only, and use hand miners for the recovery of pillars.

The disadvantage of this system lies in the fact that the advance by mechanical loaders is so rapid and the retreat by hand miners so slow that most mines soon have far too much open area with pillars that will have to stand too long before their recovery is started.

Because of the difficulty in safely and completely extracting pillars in coal seams of 15 to 30 ft. in thickness by mechanical loading, these thick beds become a curse rather than a blessing unless the operator can satisfy his conscience with only a two-thirds recovery.

The advent of mechanical loading in the Utah mines soon brought about many changes in the operations other than loading.

Power drills were brought into service. Larger mine cars and locomotive haulage to the working face immediately followed. This brought about a concentration of electrical machinery at the face, which created a decided hazard.

To overcome this hazard, line brattices are kept up to the faces of each working place, the places are rock dusted to within 50 ft. of the face and water is used extensively at the face, both on the cutter bars of the machines while cutting and on loose coal while it is being loaded. A change is being made so that all electrical equipment at the face, in gassy mines, will soon be of the permissible type. Changes and improvements in the installation of all electrical circuits are being made. It will readily be seen that a considerable portion of the savings made by mechanical loadings are immediately spent to overcome the increased hazards brought about by this type of loading.

The writer believes, however, that the preventive measures have been put in effect at a more rapid rate than the change in methods has created risks, and that on the whole the mines are safer than before these changes took place.

If this is not the case, at least one condition leading towards greater safety has been brought about by the change in methods, in the reduced number of man-hours of exposure, due to the greatly increased tonnage per man.

The change in loading methods soon brought to the realization of the operators that a different and improved type of preparation plant on the surface was necessary.

Within the past four years eight of the mines have been equipped with modern all-steel preparation plants which enable them to ship a much better product both as to cleaning and sizing.

Most of these plants have facilities for reducing to proper size the enormous lumps loaded by the machines, and ample space for hand picking the sized products.

Up to date none of the mines have found it necessary to install mechanical cleaning plants for the smaller sizes. This is due to the fact that the coals are generally quite clean, and where bands are encountered they are so hard that they go into the larger sizes when broken.

After seven years of working and experimenting, all the mines in the state that are using mechanical loaders successfully are using a mobile type loader mounted on wheels or caterpillars, and are using the room and pillar system in some form.

It is worthy of mention that all attempts to get away from rooms and pillars have met with failures, while all installations using the room and pillar methods have been successful.

One of the methods which was designed to get away from the room and pillar system and which resulted in a mechanical loading failure was a plan of development where a series of single entries, 125 ft. apart, were driven to the dip off a level entry to a distance of 500 ft. From one of these single entries a 45-degree slant was driven to the next entry on either side and a V-type long face was brought back uphill. This plan worked well until after the first major cave in the open territory, but from then on the caving of the roof could not be controlled, and all or a portion of the face was lost so many times that the method was abandoned.

In this plan, as in most types of long-face workings, the attempt was made because it offered a simple and inexpensive means of haulage and sufficient coal in front of the loader for a day's run without moving.

At several mines in the state where the coal is 5 ft. or less in thickness, and too low for the use of the mobile type loader, scraper loading has been tried with a fair measure of success when cost of loading alone is considered.

The scrapers have been used only to recover pillars which were left after hand loaders or other types of mechanical loaders had worked out the rooms, and no real efforts have been made to develop a territory for exclusive scraper loading.

With a hard floor and good roof the scraper will load coal in thin seams at low cost, but in the efforts to have a full shift supply of coal in front of it, small stumps or pillars are often left, being considered too small to justify a change in sheave location or being near the middle of a large open area, where the chances taken to make recovery are too great, and these stumps sooner or later throw weight or cause a ride onto the retreating faces.

Because of the weight due to large open spaces and the tearing and crushing action of the scraper bucket across the loading face, the percentage of small-sized coals is noticeably increased with scraper loading. In a district where small sizes are a drug on the market and bring a very low price, the savings in cost of loading by using the scraper are more than lost in realization on account of this increase in percentage of small sizes.

One of the mines which has coal 7 to 9 ft. high, with good roof conditions, has been quite successful with mechanical loading producing two-thirds of its output. In this mine roof conditions, overburden, and height of coal are such that some of the pillars can be recovered successfully with the loading machines.

This mine, with 66 percent machine loaded coal, has averaged 9½ tons per man for every man on the pay roll, both below and above ground, for the last three years.

Pillars are worked with the loading machines only during the winter months, when they can be recovered with steady time and rapid extraction. The pillars are worked out retreating by driving crosscuts 25 to 30 ft. wide through the pillar to within one cut of breaking through, and a thin pillar, 6 to 8 ft. thick, is left next to caved area as each new crosscut is started. Most of this thin pillar is loaded out by the machines when the crosscut is finished, and the balance is recovered by hand. These thin ribs

(*Continued on page* 30)

* Superintendent, Liberty Fuel Company.

STRIP MINING in the Southwest

By G. E. Nettels*

THE history and development of strip mining has indeed been an interesting one. From the wheelbarrow stage, we have witnessed a gradual growth from the team and scraper to the small semi-revolving shovel, to the larger full-revolving steam shovel, and today to the mammoth electrical excavators uncovering coal in the different sections of the country.

The industry probably found its inception in the Pennsylvania district, where wheelbarrow runs and dumps are still discernible on the anthracite outcrops.

About the year 1877, a small semi-revolving shovel was being employed in the process of uncovering shallow coal in the southeastern Kansas district. This was rather an ingenious machine and similar in many ways to the present efficient stripper. It carried a small bucket and actually uncovered some coal on its particular operation.

In 1910, a number of small, full-revolving machines were in operation in our district. This is probably about the date mechanized equipment actually assumed importance in the strip mining industry on any sort of a scale. Three or four years later, the development of the larger steam shovels became an actuality. At this time the 6, 7, and 8-yd. equipment was introduced. In the next decade there was little development of the excavators, but from 1924 on to the present date there has been a phenomenal advance in the development and design of the electric stripper.

Today we find engaged, in various sections of the country, electric strippers with dipper capacities ranging from 10 to 18 cu. yds., and one machine capable of carrying a 20-cu.-yd. bucket, with the proper combination of boom and dipper handle. They have increased in weight from around 200 tons to something in excess of 1,600 tons. Great progress has been made in the method of moving these machines. Some of the first were carried on trucks, operating on narrow-gauge track, they being unwieldy and difficult to handle. At this time, however, we find these strippers carried on caterpillars, of sufficient bearing and stearing devices to take them any place smaller machines can go. This has been one of the biggest contributions and one of the greatest time savers as regards the development of these large machines.

The Pittsburg and Midway Coal Mining Company has been fortunate in being placed in the position of experimenting with the first 10-yd. electric in the early part of 1927, and, two years later, the first machine with 16-yd. bucket, finding both experiments profitable.

The past two years have seen additional improvement—a counter-balancing feature has been incorporated, a design which tends to increase bail pull and smooth out the peaks to which the electrical machinery is subjected. As we look at these strippers today, we wonder wherein the next improvement will take place and, hastily, come to the conclusion it is difficult to conceive whereby they can be made more efficient. Perhaps the answer to future development and growth will be found in the development of conveying machinery which will tend to shorten the digging cycle of the present-day machine. A small part of the present-day operation is actually spent in the actual digging of the material overlying the coal and a far greater part is spent in the disposition of this material.

Inasmuch as strip mining is being carried on in so many districts, under such varying conditions, it is rather difficult to make any general comparisons. With the short time available, it seems the writer should make but few general statements and confine himself to the area with which he is the most familiar, that of southeastern Kansas and southwestern Missouri.

Perhaps it would be well and interesting to give you an idea as regards the geology of our particular section. Looking at a typical geologic column of the district, there will be found five workable seams of coal lying in the Cherokee shales, three of which are persistent. The bottom and most important seam is known as the lower Weir-Pittsburg, commonly called the Cherokee, which ranges from 28 to 48 in. in thickness. Overlying this, some 80-odd ft., is found the Lightning Creek seam, from 18 to 24 in. in thickness. Immediately above this is an 18-in. bed, known as the Prairie seam. Approximately 20 ft. above this is the Pioneer seam, averaging about 16 in. in thickness, and the top member, 12 or 15 ft. higher, known as the Sunshine seam, 12 in. in thickness. These five beds have all been worked at some time or another and at the present time we see our development working on the three principal seams, with Cherokee at the bottom, the Lightning Creek in the center, and the Pioneer well toward the top. In general, the overburden over these three is similar, although a hard calcareous cap rock is frequently found over the two thinner seams. The outcrop line of the three persistent seams is approximately parallel, their presence being dependent on the elevation of the terrain. The character of the material overlying the Cherokee seam can be classed as easy, fast digging, although probably a third of it has to be shot before it can be economically handled.

There used to be, in this district, a general rule that 12 in. of overburden could be stripped for 1 in. of coal. This general rule, however, has gone by the wayside and we now are stripping coal that carries a ratio of 18 in. of overburden to 1 in. of coal, and a few isolated cases of where it has gone to a ratio of 24 in. to 1 in. This resolves itself into a condition where we are encountering ratios of 1 ton of coal to 12 cu. yds. of overburden and upwards, going as high as 1 to 20. It can readily be seen that the strictest economies must be effected if such a quantity of dirt is to be removed in the recovery of a single ton of coal. It means the strippers must perform, day in and day out, with a minimum loss of time and a constant watchfulness exerted in the direction of anticipating and preventing breakdowns.

These excessive ratios will result in calling into action drag lines, working in connection with the strippers, to increase the yardage outputs at the mines where the ratios are extremely high, as it has done in other sections, principally southern Illinois.

In the past few years much has been done in the direction of preparing the overburden, where it is imperative it be loosened, in order that it may be easily moved. The introduction of liquid oxygen explosive has effected some appreciable economies in the Indiana and Illinois district, where the strata of hard rock are most prevalent.

There has been considerable experimentation done with the horizontal drill. W. H. Stewart, of the Central Indiana Coal Company, has devised a machine to drill into the high wall horizontally, and reports favorable results. In our own district, with one exception, we are relying on the vertical hole, using relatively fast black powder and dynamite.

Throughout the country, many different beds are encountered in strip mining, and each presents problems of its own. As we go from section to section, we find different methods being employed in the loading of the coal. In Illinois and Indiana we find them loading out of relatively narrow pits, with the shovel carrying a wide berm, which permits of the continuous hauling through of the coal trains without serious interruption.

In these same districts we find the small standard shovel being employed as the loading unit, with 2 to 3½ cu. yd. buckets. In some of the lignite areas, larger machines are being used as the loading medium and, in a few cases, machines with 6-yd. dippers are loading coal out of these thicker lignite deposits. In our own district we use a machine, peculiar to our locality, known as the horizontal thrust loader. This machine is identical to the standard revolving shovel in every respect, with the exception of the boom and dipper handles. This machine carries a movable boom, hinged at the foot, which allows the bucket to be placed flat on the fireclay and thrust horizontally by rack and pinion into the coal. We are thus able to remove the coal from a wider pit. While this does not affect the loading process advantageously, it does reduce the angle of swing of the stripping unit.

In the main seam we have what are colloquially known as horsebacks or clay veins, which in places literally infest the coal, running in all directions without regard to pattern. The presence of these clay veins or horsebacks accounts for the development of the horizontal thrust loader, it being deemed essential to the proper cleaning and removal of coal from the bed immediately adjacent to these

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* General Superintendent, The Pittsburg & Midway Coal Mining Company.

Semi-Automatic Lubrication of Mechanical Loaders

By A. J. Ruffini*

IN THE operation of a mechanical loading mine, many problems present themselves and among the first ones is the problem of lubrication. Proper lubrication becomes more involved if multiple shifts are operated. The mechanical loading program was started at our operation on a 100 percent scale three years ago. The method for lubrication adopted at that time was what we termed the old hand method.

In this method, the mechanical loader operator and helper lubricated the loader at the start of the shift. This required a half hour or about 6 percent of the available eight-hour shift. This method was very slow and wasteful and in many cases a bad job of lubrication was the result, since the men on these units are working on a tonnage basis. The result was numerous burnt bearings, long delays and high maintenance cost. Discipline was resorted to in trying to correct this, but even this did not entirely insure the management of a good job in lubrication. A loophole in the discipline was the multiple shift for it was hard to pin down the responsibility.

The oil and grease wasted was quite alarming for in the old hand method, a barrel of grease and a barrel of oil was supplied to each working section. This meant 18 barrels of oil and 18 barrels of grease distributed all over the working area. The operator and helper were supplied with a hand pressure gun and two five-gallon oil cans. The operator with the aid of the gun, lubricated all bearings (32) and the helper filled all gear cases (7). Even with the utmost care, there was considerable oil and grease wasted not only while lubricating the loader but also in filling the hand gun and cans.

It was very evident that a different method of lubrication had to be adopted in order to reduce mechanical failures and to reduce lubrication cost. The management decided to lubricate the loaders outside of regular working time with a crew of men that would be responsible to the chief mechanic. This method would increase the productive time about 6 percent and would insure a better job of lubrication. This system was used for about six months and the results obtained were better than the old hand method, but it still meant lubricating the loaders with the aid of oil cans and a hand pressure gun. It still meant oil and grease was being wasted and stuck bearings were not eliminated as it was impossible to flush out bearings with the hand gun. The problem resolved itself into securing or building a semi-automatic lubricating machine in which all the handling of oil and grease would be done semi-automatically. Numerous letters were written to lubricating equipment manufacturers, but nothing any ways near our idea could be obtained. It meant that we had to design and build a semi-automatic lubricating machine. After several months of planning and designing, we built a machine that will handle the lubrication of our loaders semi-automatically even to the filling of the oil and grease tanks on the machine from the drums.

The construction of the machine is as follows: The bed consists of the standard bed for our mine car, 3 ft. wide by 12 ft. long, constructed of 2½-in. plank with steel reinforcement. This is mounted on a 42-in. gauge, 28-in. center, roller bearing truck. On one end is mounted a 75-pound capacity electrically driven lubrication gun equipped with a metal braid, rubber cover, high pressure hose. The gun builds up a pressure of 3,300 pounds which makes it very easy to not only lubricate but also to flush out all bearings. In the center of the machine is mounted two 78-gallon capacity tanks. One for oil and one for grease. This capacity was governed by height of seam and length of truck bed, but it is more than sufficient to lubricate all loaders on the machine's territory. On the other end of the truck is mounted an air compressor with a capacity sufficient to maintain a pressure of 80 pounds on each tank at all times while the machine is in operation. The pressure is controlled by an automatic unloading valve which allows the compressor to run continuously while lubricating the loaders. The oil tank is equipped with 24 ft. of high pressure, rubber, oil hose and an oil flow meter. The grease tank is equipped with 30 ft. of high pressure, rubber grease hose and also a flow meter. All piping consists of flexible metal hose to take care of all shocks.

In filling the tanks, the oil and grease drums are taken into a special room located near the workings called the filling station. Here the drums are placed on a rack. The compressor is started on the lubricating machine and an air hose is attached to the ¾-in. opening on the drum and a grease hose is attached to the 2-in. opening and run to the grease tank on the lubricating machine. This provides rapid and complete transfer of oil and grease from the drums to the tanks with no waste. Under this method, the oil and grease is handled by tank and hose from the refinery to the bearings and cases.

The actual lubrication of the loading machines is done by a crew of four men, two for each lubricating machine between shifts. We operate two eight-hour shifts with 3½ hours between each starting

* Efficiency Engineer, The Wheeling Township Coal Mining Company.

Greasing machine with hose in place ready for transportation

The oil hose, grease hose and hose for high pressure bearings

Form No. 121-M-10-30-M

....193......

OIL AND GREASE REPORT

JOY NO.	METER READING	GALLONS OIL	METER READING	GALLONS GREASE

FORM 1632 — TIME AND MOTION STUDY RECORD — Mine NO. 2

OPERATION Lubrication of Loaders — OPER. NO. — SERIAL NO.

COMPILED BY Buffini — DATE 4/27/31

DETAILS	UNITS OF OBSERVATION						CALCULATION OF UNIT
SHEET No.	1-1	1-5					Per Loader
Travel to loader section	7.00						7.00
Put cable reel nips on	.50						.50
Travel to loader	1.50						1.50
Start compressor motor on lubrication machine and start loader motor	.50						.50
Remove grease, oil and gun hose	.75						.75
Grease bearings & chain (One Man)	8.00						8.00
Fill grease cases (One Man)	(6.50)						(6.50)
Fill hydraulic oil tank (One Man)	1.50						1.50
Grease armature shaft (One Man)	(3.00)						(3.00)
Place all hose on truck and stop compressor	.75						.75
Check grease and oil meters and record readings	.50						.50
Fill lubrication gun tank		2.00					.40
Return lubrication machine to station		7.00					1.40
Clean up machine		10.00					2.00
			Total				24.80
			Total	Man Minutes			49.60
UNITS OF OBSERVATION							
UNITS OF CALCULATION							

time. The lubricating crew start their shift at 10:30 a. m., which is 3 hours later than the regular shift's starting time, and quit at 7 p. m. Two of the four men's first duty is to service the lubricating machines, seeing that they are ready; this takes about 1½ hours of their time. The balance of the time they work on pulling track and post from finished workings. At the end of the regular shift, each crew secures one of the gathering motors. This motor is hooked onto the lubricating machine and the machine is moved to the different sections where the loading machines are working. The time study on the actual lubrication of the loading machines plainly shows the cycle and the speed with which the lubrication is accomplished. Each loading machine has 32 Alemite connections, 7 gear cases and one hydraulic tank that needs attention every shift, and two armature bearing cups that are filled every other working shift, but are inspected every shift.

Cost records on lubrication are easily kept on each individual machine as each lubricating outfit is equipped with meters, one for oil and one for grease. The greasing machine operator records, on a card provided, the readings of each meter after lubricating each machine. This card is sent to the Cost Department where the amount of oil and grease used is charged to the different loading machines. The amount of grease used in lubricating bearings is divided among the machines since no accurate check can be obtained on this other than that the 75-pound tank is filled when needed.

Mining Systems in Utah

(*Continued from page 27*)

have been found very effective in holding the roof over the working places while the bulk of the coal is being removed.

At this mine rooms are driven on the strike 600 to 1,000 ft. long. Double tracks are laid in each room close together on the low side of the room, and since the loading machines travel on caterpillars from room to room through the nearest crosscut, a removable section of track is placed opposite each crosscut so that, though the machine is sometimes climbing a 10 percent grade through these crosscuts, it has no trouble doing so, as would be the case if it had to cross over a track while on this pitch.

At one of the mines which has coal 26 ft. in thickness and is very gassy, the method of working is as follows: There are four main entries, two intakes and two returns driven on the strike of the seam. Three entry panels are driven half across the pitch on 5 to 8 percent grades with rooms on the strike. Rooms are driven off both sides of the panel, using the room entries for intakes and the middle entry for the return. The rooms are driven 32 ft. wide on 84-ft. centers. All haulage is with locomotives and 5-ton steel cars.

The first mining is done with small shovels, taking the lower 8 ft. of the seam, in rooms driven full width to a distance of 500 ft. When the rooms are completed the 16 ft. of top coal is shot down the entire length of the rooms and then loaded out with a larger type shovel. After this top coal has been loaded out a skip or slab is taken off the upper side of the rooms, reducing the pillars to 45 ft. in thickness. No further attempt is made to recover the pillar, and when a panel is worked out to this extent it is sealed and a new one started.

This 26-ft. coal seam has overburden ranging from 900 to 1,500 ft. in thickness.

Six-ton battery locomotives and 13-ton gathering and main-haul locomotives are used.

Permissible mining machines only are used, and all machine operators carry a safety lamp and inspect for gas before starting to cut.

All shot holes are bored with electric drills. The places advancing are cleaned of bug dust immediately after being prepared for shots.

All places in the mine, both those on the advance and pillar sections, are rock dusted to within 15 ft. of the face, redusting being done by machine each week.

A number of costly lessons have brought to the attention of the operators of this district that if mechanical loading is used, more than ordinary precautions must be taken along with it, and if these extra efforts are not made the mines are better off worked on a hand-loading basis.

GATHERING SYSTEMS
with Mechanical Mining

By C. J Sandoe*

At THE time of the installation of loading machines at Taylor mine of the Perry Coal Company, near O'Fallon, Ill., the company had been using small mine cars and mule gathering. To get a reasonable tonnage from this mine with a loading machine without abandoning or discarding this equipment developed a somewhat serious problem. All of you are familiar with the fact that any reduction in tonnage increases materially the production cost per ton of coal, and that delays due to inefficient haulage or accident in the haulage road are probably responsible for tonnage losses more than any one factor about the mine. It is obvious that a plan had to be worked out that would involve a rapid car change and one that was as simple as possible.

In doing this several factors had to be taken into consideration. There was the arrangement of the haulage system so as to insure an ample supply of empties each minute of the operating day and the keeping in condition of the track and equipment in use. Lost loading time with a loading machine can not be made up. Cars loaded mechanically do not load as heavily as cars loaded by hand. For example, the mine cars at Taylor mine carry about 2,100 lbs. of coal loaded mechanically, while in hand loading these same cars have an average capacity of 2,900 pounds of coal. In other words, to maintain the tonnage at the mine this meant that the men had to handle approximately 33 1/3 percent more mine cars both in and out than with hand loading. The small mine cars, of course, to a degree aggravated the situation. This is something that you who are equipped with larger mine cars will not be affected by. The result was that the use of mechanical equipment had to contemplate speeding up the haulage sufficiently to offset the lost weight per unit caused by mechanical loading if the same tonnage was to be produced each operating day.

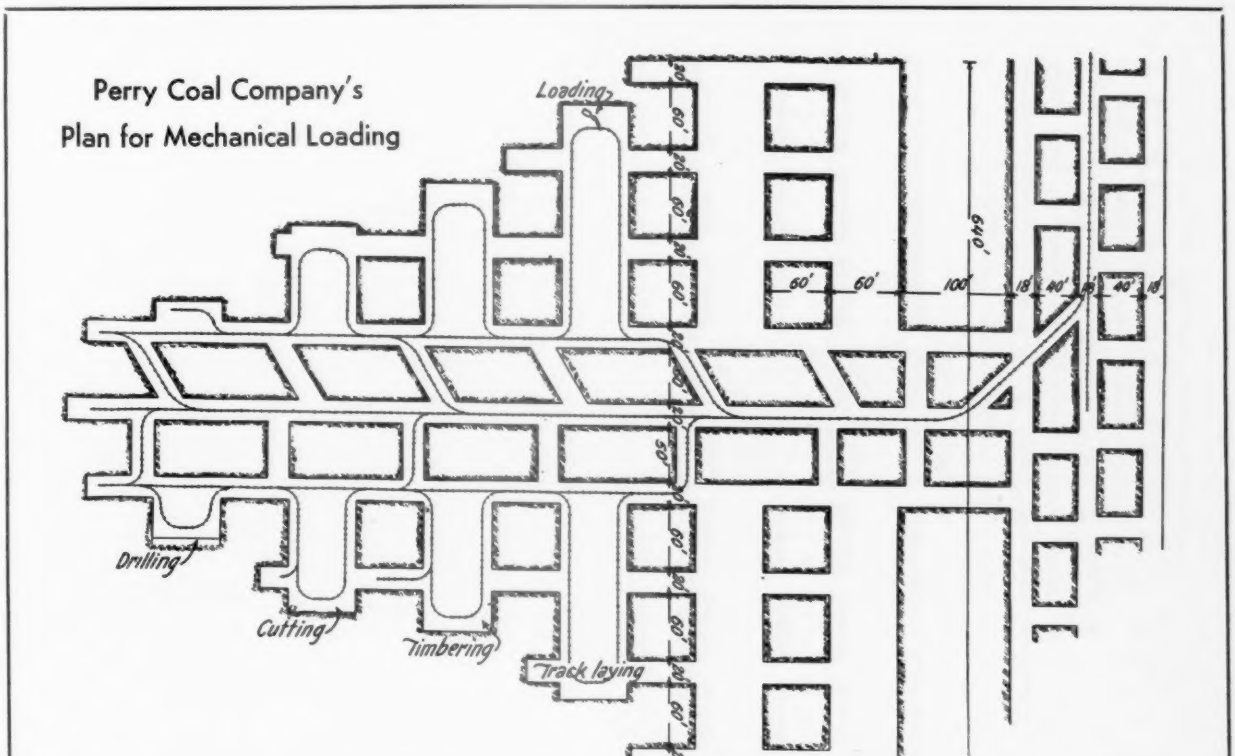
The system of gathering from loading machines had next to be taken into consideration and worked out. The system of gathering is a determining factor in getting a high tonnage from a loading machine. The fact is a time study of any loading machine during its best days will show much time lost in making car changes at the machine and in getting these cars to the motor road. Some car changes could be made while the loading machine is moving or digging tight coal, but a system had to be worked out with the idea of cutting these delays to the minimum.

The mine fortunately has very excellent physical conditions for underground loading. The seam mined is the Illinois No. 6 and averages approximately 7 ft. in thickness. Immediately under the coal there is from 6 to 14 in. of fire clay and below the fire clay is partially decomposed rock. The fire clay is dry and comparatively hard and slick. Above the coal is a thick limestone which is both hard and tough, requiring but few timbers in rooms 30 ft. wide. The limestone varies from 18 to 40 ft. in thickness.

To overcome the difficulties and to take advantage of the favorable natural conditions a modified panel three-entry system was developed. Off the outside, entry rooms were driven 60 ft. wide on 120-ft. centers to a depth of 240 ft. The rooms were not necked but were turned 60 ft. wide off the entry and the crosscuts are driven 60 ft. apart, leaving pillars 60 ft. square. Tracks were laid in all three entries. In the center entry the track was permanent for the life of the panel, while in the outside entries it was removed as the rooms were abandoned. Crossovers were placed in every other crosscut from the center entry to the

* Vice President, Perry Coal Company.

Perry Coal Company's Plan for Mechanical Loading



outside entries. This gave an added efficiency in the haulage, as the empty trips came in the center entry and the loaded trips were hauled by way of the outside entries to the parting. The outside entries had two switches for each 60-ft. room, the outside switch being reversed to keep the mine car from being turned around. The track extending into the rooms (slabs as these placed were termed locally) was laid with its center 8 ft. from the rib and continued to a point 35 ft. from the face, then curved 90 degrees on a 15-ft. radius and extended parallel across the face, curved again 90 degrees and returned on the opposite side of the room until it connected with the track in the entry. This track arrangement allowed ample operating room between the track and the face for the caterpillar loader and also allowed approximately 38 ft. timbering space between the tracks. The timbers used were 6-in. round oak props, spaced 4-ft. centers to average rock, with the necessary additional timbers where unusual conditions were met with.

If the explanation made is not entirely clear, the maps here will graphically show the method of development which has been described with the track layout and haulage, and will clarify the explanation.

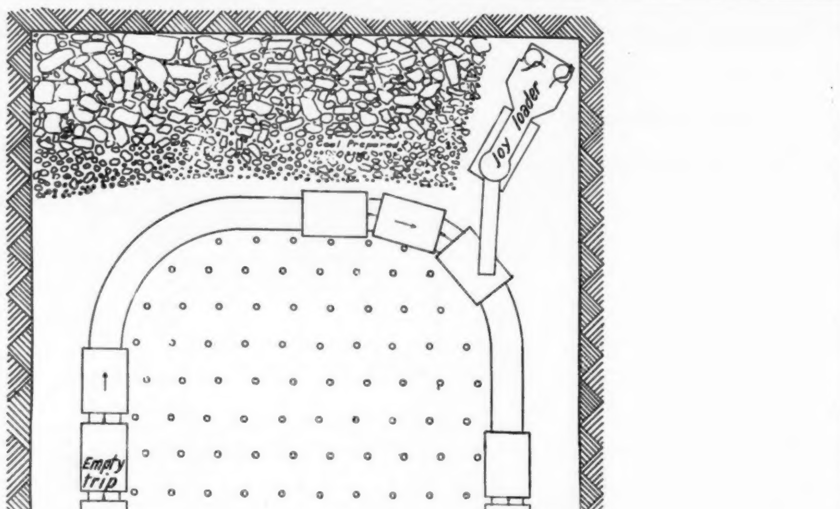
The speed of mine car loading obtained by this plan can best be impressed upon you by following a trip around the room track. For example, the first driver with a trip of four to six cars is headed out of the room by the outside switch. The second driver is at the loading machine with a trip of four to six cars. After 30 to 40 seconds have elapsed mine car No. 1 will be loaded, then car No. 2 follows, etc. The first two cars are placed in reference to the loading machines, so that the conveyor can be swung into position for loading either of these cars without moving the loading machine or the cars. As the cars are loaded, the driver moves the trips to spot the next two cars before the machine for loading, and so on, until the entire trip has been loaded. A time study of this operation has shown as high as eight cars loaded in three minutes. By the time the second driver's trip is loaded the third driver is returning from the parting and coming into the room by the inside switch, which is reversed and is in a position to pull under the loading machine as soon as the second driver has moved out.

Under the system described we have loaded approximately 350 mine cars a day. For example, the following is one week's record for a loading machine with the tonnage at Taylor mine:

	Cars	Tons
Monday	339	355
Tuesday	318	322
Wednesday	354	371
Thursday	347	364
Friday	328	344
Saturday	390	409
Average	346	360 5/6

Your attention might be also directed to the fact that in getting efficiency in the gathering operations it is quite necessary that the animals as well as the men be trained. In other words, you do not obtain the best results the first day the machine is put in the mine. The mules have to be accustomed to make short pulls and stopping, and doing this rapidly. The mules can be trained, however, and the men accustomed to the system, performing with accuracy and smoothness at the loader within approximately a month's time. To handle the mine cars from a parting to the loading machine and to the parting again for the tonnage mentioned the services of three drivers are required. The minimum distance from the loading machine face to the parting is 700 ft. and the maximum 1,400 ft. The haulage cost per ton per mule under mechanical loading is slightly higher than in hand loading, but any additional increased haulage costs are more than offset by the decrease in loading cost.

It is not believed that the experience or methods used at this mine will solve the problem of mechanical loading and haulage at every mine. This paper is not read with this object in view. Its purpose is simply to give to you this company's experience. It would seem fundamentally, however, that the gathering and removing of the coal from the loading machine is the most serious problem involved in the efficient use and operation of mechanical equipment. To that extent the problem of virtually every company is the same. In working out this problem there is involved the use of the three factors, power, track and cars, each one depending upon the other, and all companies have these same materials to work with. The use that can be made of them will, of course, by necessity, vary with the mine.



Perry Coal Company's 60-ft. Face Plan

MECHANIZATION

at the Carbon Fuel Company

By C. A. Cabell*

THE Powellton Seam on the property of the Carbon Fuel Company runs in height from 8 to about 11 ft. Three feet up from the bottom there is a band of slate varying in thickness from 8 to 24 in. While the two benches of coal above and below the slate are of a very similar structure, they have certain characteristics which are different. The bottom bench is of a very low ash and medium fusibility. The top bench is higher in ash and has a very high ash fusion temperature. A mixture of the coal from the two benches makes a low ash product with a fusion temperature higher than that of either coal mined separately. The combination of the two, due partly to the above characteristics, makes a high grade domestic coke and has a good by-product yield. The top of the seam is only a fair slate top. The bottom is a very hard slate. The coal separates very readily from both bottom and top, but the top, in some cases, weathers very badly.

This seam, until 1924, was mined by hand. The preparation at the tipple was hand picking. The bench system of mining was used. The coal was cut with middle cutting Arcwalls just below the slate band. Three holes were drilled in the top coal. The shooting of these holes brought down both top coal and slate, freeing the coal from the slate. The coal was then loaded off the slate, after which the slate was broken up to such size as could be handled. The next operation was the loading of the bottom bench. The hand method of mining this seam, as described above, presented considerable difficulty in the way of both preparation and cost of operation. With slate running as thick as this, the cost of handling by hand was necessarily very high. With so much dead work to do, the tonnage output per man was of course low. It was naturally the desire to have mine car trips contain an equal number of cars from top and bottom benches. It was also in the interest of efficiency to have slate trips hauled at regular intervals during the shift in order to avoid delays at the tipple waiting for coal. In a large group of men, all working at different rates of speed, this was next to impossible. Many times in large sections of the mine all loaders would be taking out the slate band at the same time. Often entire trips would carry only the top bench, bottom bench or slate. In order to overcome this, extremely costly shifting and much additional side track was necessary. If a trip of coal from the top bench or bottom bench were dumped separately, one railroad car would have one ash content and another a very different ash. It is well known that a large fluctuation in ash content is a very undesirable feature in the by-product market. As the competition for by-product coal became more intense, the mines which did not have these natural obstacles naturally became more profitable—the logical result being that the mine above described became less profitable. Therefore, even though the natural qualities of this coal were well recognized, it was decided to close the mine until better means could be worked out for mining and preparation.

In 1928 and 1929 a careful study was made of both mechanical mining and preparation in an effort to see how far these two developments had advanced toward overcoming the obstacles mentioned above. The result of these studies was the complete mechanization both of loading and preparation of a mine in this Powellton Seam.

The mining is done entirely in sections. A section comprises a crew of men with a section boss, a unit of mechanical equipment and a given territory.

The crew of men consists of a loader operator and helper, cutting machine operator and helper, a gathering locomotive crew of two men, a face preparation crew of two men, three men drilling and shooting, two men timbering, and from one to two track men.

The unit of mechanical equipment consists of a loader, gathering locomotive, cutting machine and a drill. Two types of loaders are used in this mine—the 5 BU Joy and the No. 4 Myers-Whaley Automat. The machines are Goodman slabbing machines and the locomotives General Electric 8-ton trolley and cable reel. Two types of drills are used—a Sullivan self-propelling mounted drill and Little Giant Electric drills.

A careful study of the size and type of mine car revealed that a car of 156 cu. ft. capacity or about 4 tons mechanically loaded would probably be the most economical in point of transportation cost. The study of upkeep resulted in the selection of the Kanawha box type car with steel sides and wooden bottom. The cars are equipped with Timken bearings and four-wheel brakes, spring drawbars and swivel couplings to allow dumping in trip in a rotary dump.

As all of the work in the mine at the present time is development, the territory for each section is composed of an entry with from three to four headings and the necessary cross cuts—making from four to five working places at all times. It was found that no less than this number could be economically driven. In the laying out or projection of these entries, three things are kept in mind; namely, ventilation, transportation during the time the entries are being driven and the transportation which these entries must later furnish from the rooms after the development has been completed. All cross cuts are driven on an angle of 45 degrees from the entries in order to accomplish rapid transportation. The section crew drives the entries on 30-pound steel with steel ties. Complete sets of all steel tie turnouts are used to turn all cross cuts. After every 200 ft. of advance the main haulway entry is layed in heavy steel on treated wood ties. The treatment of all wood is the open cell method using Wolman salts and is done in a small plant on the property. The other parallel entries which have been selected for side tracks are laid in 30 pound steel and wood ties. Track is pulled out of any remaining entries.

Approximately 18 in. of top coal is left. This coal is high in impurities and furnishes an excellent protection against the weathering of the slate top.

The cycle of operation is about as follows: Starting with a cleaned up place, the machine enters and cuts in the coal directly above the slate band. The machine cuttings are thrown back from the face by hand while the machine is moving to the next place. Three holes are drilled 18 in. below the roof and three holes exactly parallel to and on the slate bottom. Three holes directly under the slate band are then drilled, loaded and shot. The machine crew next returns, sumps into the slate band and rakes the slate out on the floor. The loader now enters, first loads the machine cuttings, which have been thrown back from the face, and afterwards loads the slate. The preparation crew then picks all loose pieces of slate out of the back of the curf and carefully sweeps it, after which the crew loads all the top and bottom holes, shooting first the top holes and then the bottom. The result is a well-mixed coal. The place is now ready for the mechanical coal loading operation.

Some experimenting has been done on the method of drilling and shooting as well as with the different kinds of powder. At the present time Gelobel is being used for shooting the slate and Monabel for the coal. The Gelobel is in small cartridges 1 in. in diameter, and the Monabel is 1-5/8 in. in diameter, so that there is no possibility of the preparation crew mistaking one for the other. A small dummy machine outside the mine makes dummies at the rate of about 2,000 per 8-hour shift for one man. These dummies are filled with clay or loam and are used for tamping. The powder and caps are brought into the mine in specially designed powder cars. These cars are placed in cross cuts a safe distance from the face. The men carry enough powder to shoot one or two faces from the car to the face in canvas bags.

In our experience, one of the most necessary features of the successful working out of mechanization, as well as one of the most difficult to properly organize, is the maintenance. All too often, the keeping of the mechanical equipment turns out to be repair rather than maintenance. An efficient maintenance system must carry proper lubrication, regular inspection, an adequate stock of parts close at hand and careful records

* President, Carbon Fuel Company.

on the cost of maintenance and repair on each piece of machinery. This latter not only is a true check on the efficiency of the maintenance system, but also on the quality of the machinery and the efficiency of the crews operating it.

Each operator is responsible for the proper lubrication of his own machinery. He carries one can for each different kind of lubricant necessary with him as he goes on shift.

For the four sections (each double shifted) three mechanics and a chief electrician look after the upkeep. The mechanics are on duty 24 hours a day—one for each 8-hour shift. Their duties include some time spent watching each piece of machinery in operation and talking to the operator in order to learn of any performance not up to standard of his machine.

A supply house carrying a complete stock of parts is located at the mine mouth. All parts are kept in separate bins and a Cardex system of continuous inventory is used. Al parts and lubricants are issued on requisition. The supply clerk then extends each requisition and posts it against the record kept for the machine to which the parts are issued. Each mechanic turns in a time card at the end of the shift to the supply clerk showing the exact time spent on either inspection or repair for each machine. The supply clerk makes a monthly report on the upkeep from each section.

All of the above is necessary to put the coal to the gathering partings. The organization from there out is very similar to that used in any coal mine, with the exception of the care in planning and the proper upkeep of the main haulage system. As a very great part of the success of a mechanical loading installation depends upon rapid transportation, it is extremely necessary to minimize the time in which the gathering locomotives have to spend waiting for cars. The main haulage in this mine is about 12,000 ft. long, with the grade in favor of the loads. Sixty or seventy-five pound steel is used entirely for this haulway. This steel is placed on 6 in. x 6 in. x 6-1/2 ft. treated wood ties well ballasted. All posts, caps, headers and wedges are of treated wood.

One of the main power intakes into the mine follows this main haul road. Every rail is double bonded, with a "U" bond on the outside and a straight bond on the inside. The track is cross bonded every five rail lengths. No. 9 section trolley wire is paralleled with the 500,000 C. M. feeder. The entire main haul road is illuminated and man holes are placed every 80 ft.

An 18-ton General Electric armor plate and a 15-ton Jeffrey armor plate comprise the haulage locomotives. At the mid-point of the main haulway the road is double tracked for a distance of 1,600 ft, with three crossovers. This allows the haulage locomotives to pass as each goes from the gathering partings to the headhouse, or it allows one locomotive to travel to the half-way point and change trips with the other. It is planned, at a later date when the development has advanced sufficiently to allow higher tonnage, to place a dispatcher at this mid-point so that the moves of the haulage locomotive from one inside parting to the other may be minimized. The coal and all slate from entries is hauled to the headhouse. Both coal and slate are dumped in an electrically driven rotary dump. By means of a fly gate coal is diverted into a large storage bin and the slate into a small bin located at the loading terminal of an aerial tramway. This aerial tramway carries the slate and cleaning plant refuse to a point 3,600 ft. over the hill and automatically dumps it into a large hollow.

A natural corollary to the complete preparation of a mechanically loaded coal is a cleaning plant. This mine is equipped with a Fairmont-Peale-Davis plant which air cleans the entire product at the rate of 250 tons per hour.

The two chief conditions which govern the success of 100 percent mechanization are: *first*, the quality of the product resulting from the mechanized mining, and *second*, the cost of operation by this system. In this mine it is being demonstrated that the average ash content of the entire product is lowered about 3 percent under that obtained under the hand mining system. The uniformity may be judged by the fact that the maximum fluctuation between any two railroad cars seldom exceeds fourtenths of 1 percent. This, in spite of the fact that the coal in one railroad car in hand mining, comes from 12 to 15 working places and that from mechanical mining comes from never over three places. As to the success from a cost standpoint, the best comparison is on a tons output per man basis. The yield in the hand mining system was about 5-1/2 tons per man shift. The present yield for the first three months of this year, after only nine months operation, is 7.7 tons per man. The latter figure for mechanized mining is purely on development work, while the hand mining figure is for work in both rooms and entries, which naturally gives a higher output per man.

It has been truly said that the efficiency of mechanical loading is dependent about 10 percent on machinery and 90 percent on management. With more experience and when a large portion of the tonnage is coming from rooms, much better results are expected.

Strip Mining in the Southwest

(*Continued from page* 28)

clay streaks. To eliminate impurities and to effect loading economies, our company two years ago installed a small revolving shovel with a bucket 16 in. in width, which is used in the removal of these clay impurities. It readily can be seen that more time can be spent in the actual loading of coal from the bed, with a resultant increase in tonnage, when these clay veins do not have to be contended with.

A few operators in our district are using the mining machine, working in a vertical position, to cut the coal immediately adjacent to and parallel with the track, and at other points across the pit. These machines, to be more explicit, are identical to those used in the shaft mines where the coal is undercut, except instead of operating flat, they are inverted, with the cutter bar operating vertically. Built on a skid, they are propelled by the chain in the same fashion they are moved in the underground mine. The advantageous effects gained, from the operating of these machines, are a protection to the track, a decrease in screenings, and a saving in the amount of explosives necessary to prepare the coal for loading. The only objection we have ever encountered is that it frequently makes lumps so large they will not drop out of the haulage equipment. Constant surveillance is necessary to preclude this possibility.

From the graphs in the excellent pamphlet, "Economics of Strip Coal Mining," published by the U. S. Bureau of Mines in 1931, some interesting information is available. We find, as regards output per man-day, throughout the strip operations, tonnages ranging from 37 at a property with a ratio of overburden to coal of less than 2.9 down to less than 4 tons per man-day where the ratio is in excess of 16 to 1. In our own district a fair average would probably be in the nature of 12 tons per man-day with a ratio of 14 to 1, with a maximum of 17 tons per man-day with a ratio of 16 to 1. It should also be noted in connection that the average price f. o. b. mines at this extremely high ratio mine was around $2.60. The average price f. o. b. mines, taken from the same graph, is around $1.65 per ton for the industry as a whole.

One other item in connection with strip mining of our district that differs possibly from that of other is the method of haulage employed in moving coal from the pit to the preparation plant. In most strip mines the track is laid on the coal and the haulage system built up on that basis. In a few isolated cases in our district the coal is being hoisted out of the pit by means of a crane and skip, to cars on top of the bank. At one of our company's mines we are loading and hauling on this method, using standard-gauge rail which, instead of being torn up from pit to pit, is merely skidded over with the aid of a tractor. This one operation has materially reduced the track costs at this mine over the old method of building and tearing up from pit to pit. Naturally, the topography must be readily adaptable. Gently rolling prairie is a requisite for this sort of system. It would be out of the question in a hilly or severely rolling country, as is often found in strip-mining territory.

With a high percentage of strip mines operating in our territory, the operations have become seasonable in nature. This probably is true in many cases in other districts, the main stimulus to business being "Old Man Winter." We, like many others, are confronted with the problem of being able to supply a demand during peak periods and to hold our overhead and investment expenses down to a minimum during the dull periods. The ratio of fixed charges to variable charges is one that gives every management much to think about. After all, the best sort of an intelligent average has to be arrived at and efficiency exercised if strip mining is to continue profitable.

Mechanical Loading at Little Betty Mining Corporation

By P. L. Donie*

THE Little Betty Mine of the Little Betty Corporation located near Linton, Ind., is mining the No. 4 seam which is at an average depth of 240 ft. below the surface. This seam of coal in this locality is typical of the No. 4 seam, which is of a rather hard, firm structure. The average thickness is approximately 6 ft. This mine has been in operation for 11 years and the working face is now approximately 1¼ miles from the shaft bottom.

The room and pillar system of mining is used at this mine. The rooms are driven 26 ft. wide and 200 ft. deep on 34-ft. centers. The entries are cut to a width of 12 ft. The track is carried in the center of the rooms. There are three main parallel entries from which are turned panel entries. The rooms are driven from these panel entries. Sixty to 70-lb. steel is used in the main haulage ways, 30-lb. steel in the cross entries, and 20-lb. steel in the rooms. Roof conditions are moderately good. As a rule, three rows of posts must be set on each side of the track in rooms.

A few years ago this mine was practically all hand loading. An investigation of the practicability of pit car loaders was made and we found that a material saving resulted by the use of these over hand loaders. The output of the mine was increased, and work concentrated. This resulted in a decrease in the cost per ton of coal mined, over that of hand loading.

The pit car loader was used for a number of years with satisfactory results, but there was a general dissatisfaction against them on the part of the miners. It was at this point that we decided to go further into mechanization by investigating the possibilities of loading machines for this particular mine.

Loading machines of the large, mobile type were first considered. We found that this type of loader would not be suitable since they were not adaptable to our conditions. There are two outstanding conditions in this mine that would be against the successful operation of a larger loader. The first is that the cars in use are small, having a capacity of only 1½ tons and, second, the posts have to be carried near the face, to control the roof. Taking the above into consideration it was decided that any investment in large machines would not be warranted as it would be impossible to get the average capacity from this type of machine to make them profitable.

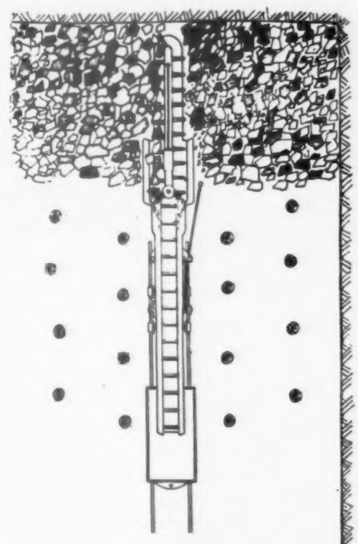
We decided to try out the Jeffrey 44-C loading machine since it seemed to be ideal for our purpose, both from the standpoint of investment and capacity. With the same investment in loading machines, that is, three 44-C machines as against one large machine, three cars can be loaded simultaneously. Therefore, with the 44-C, the small cars are not much of a handicap; as a matter of fact, in our case they are rather an advantage as the men can switch them by hand.

The loaded cars are pushed down the straight-away where they are picked up by a gathering locomotive and taken to the parting. The switches are carried near the face so that the cars will not have to be shifted any great distance.

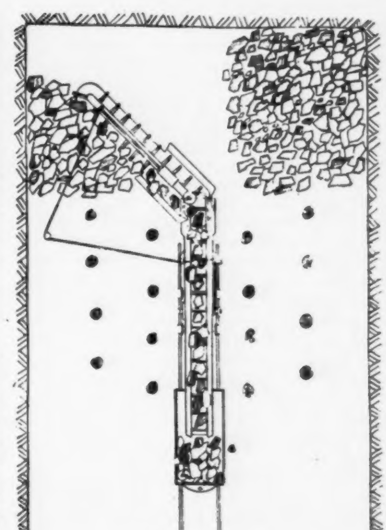
The usual practice is to drive two cross cuts through the rib on each side of a room and stagger them, and provide all four with a switch. The average time for shifting cars is 1¼ minutes. However, in some cases the cars are shifted in as little time as 30 seconds.

The 44-C machine is composed of two main elements; a discharge conveyor which is permanently supported on a truck, and a gathering conveyor which is carried on a pony truck while the machine is being moved from place to place. Both of these conveyors are pivoted on their respective trucks, for which reason the machine can negotiate

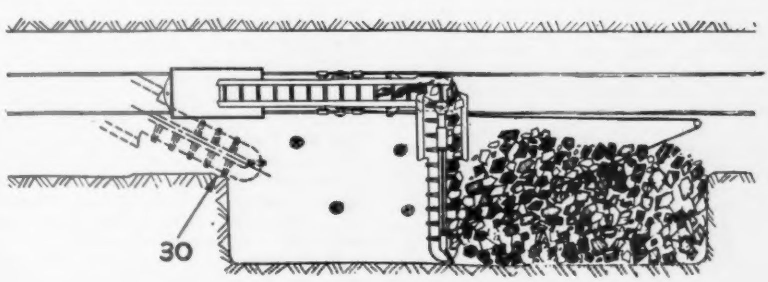
(*Continued on page* 52)



SUMPING



LOADING LEFT SIDE OF ROOM.



SLABBING

* Vice President, Little Betty Mining Corporaiton.

ANTHRACITE RESEARCH for Utilization

By C. A. Connell*

RESEARCH is not new to the anthracite industry, although it is only within the last few years that the anthracite producers as an industry, have conducted joint research into the utilization of anthracite. The methods now in vogue in the mining of anthracite and the huge modern breakers which have been erected in the anthracite region are monuments to the extensive research which has been conducted over the past hundred years in connection with mining and preparing anthracite.

Individual producing companies have also contributed materially to the development of anthracite consuming methods through research in the boiler plants at the collieries and through investigations which have, in more recent years, been conducted at laboratories on various companies' properties. Establishment of the industry's laboratory at Primos, Pa., last May is, therefore, a logical development to coordinate, expand and thereby make more effective a program of research which had its inception very shortly after anthracite was first discovered.

At the present time the research being conducted by the Anthracite Institute is divided into two main classifications. Research at Primos is primarily concerned with the study of methods of burning anthracite, with the evaluation of anthracite consuming and regulating equipment, and with the further development of such equipment. The Institute's research at universities is of a more academic nature, including the determination of the fundamental constituents of anthracite to the end not only of developing facts which will permit of more intelligent study of methods to improve the utility of anthracite as a fuel, but also to provide the basis for the evolving of entirely new uses of anthracite outside the fuel field.

Prof. Homer G. Turner, of Lehigh University, Director of University Research for the anthracite industry, placed as his primary objective the finding of the answer to the question, "What is anthracite?" To that end he has studied a large number of samples of anthracite from different regions and different beds under the microscope and through the X-ray. This study will be continued until anthracite is completely catalogued, not only chemically but also organically, for future and more definite phases of investigation.

Professor Turner was one of the first geologists to discover the adsorptive power of anthracite. By "adsorption" is meant the adhesion of the gases or dissolved substances to the surfaces of solid bodies resulting in a relatively high concentration of the gas or solution at the place of contact. Materials having this characteristic are, as you know, demanded by many different industries. The basic determination of the constituents of anthracite will make possible the more intelligent selection of the relative absorptive powers of the various types of anthracite, and it is confidently expected that new markets will, therefore, be found for such selected types of anthracite. Professor Turner further plans to experiment with various activating agencies to increase the absorption of anthracite. He has discovered other characteristics of anthracite which may have important commercial value. Among those discovered is the present use of small-sized anthracite in filtration beds of more than 100 water purification plants. He has set up a series of filters to compare the value of small-sized anthracite with the sand and gravel mixtures which are more commonly used. Bacteria of the type found in poluted waters are passed through the filters and it has been determined that fine anthracite is much more efficient than sand and gravel in the removal of bacteria. He is further comparing the relative efficiencies of anthracite with sand and gravel in the removal of odors, color, and turbidity.

At Penn State College research for the anthracite industry is being conducted in connection with mine shale, which, it has been found, when pulverized, mixed with water, pugged and subsequently heat treated and dried, may be made into blocks having a greater compressive strength than concrete. The findings to date suggest the possibility of the use of those blocks for mine props, construction purposes, etc. One of the largest pottery companies, after reviewing a report of these preliminary investigations, has requested larger samples of the shale for experimentation at their own plant in the manufacture of clay products.

The commercial value of academic research is, of necessity, indeterminate in its early stages. It is for that reason that we feel highly encouraged in the short space of time in which the products of the anthracite mines have been under study, that several attractive commercial possibilities have already been presented for further development.

At Primos, Pa., 8 miles from Philadelphia, the Anthracite Institute has set up their laboratory, under the direction of Mr. A. J. Johnson, for the purpose of providing maximum service to inventors, equipment manufacturers, retail coal merchants and consumers who are interested, from their respective points of view, in the merit of anthracite consuming devices.

The anthracite industry has interpreted research for utilization to mean research to provide the widely varying types of heating comfort which consumers demand. Some consumers are primarily interested in the cost of heating their homes. Others are interested principally in freedom from manual attention. Those are the two extremes, and between them come the vast majority of consumers whose desires are a combination of those two in varying proportions.

The Anthracite Institute Laboratory is housed in a 120-ft. by 40-ft. brick and concrete building, constructed for testing purposes, boilers used for test purposes were selected to cover the range of boiler sizes from the 18-in. vertical round to the 50-hp. steel tubular, thus covering the boiler field from the smaller homes to medium-sized apartment houses.

The latest approved testing instruments, including recording CO_2 and draft gauges, recording pryometers, etc., have been permanently installed on panel boards remote from the boilers. The arrangement allows changes of instruments from one set of boilers to another by merely changing couplings, and saves time, minimizes errors, and permits the observation at one point of two or more comparative tests when run concurrently.

The feedwater-pipe arrangement is typical of the precautions which have been taken throughout the laboratory against error. The water is metered, and then passes to a tank where it is weighed; from that tank it is pumped to a measuring tank above the boiler being tested, thus providing a three-way check.

The recording instruments are periodically checked by instantaneous reading instruments and the coal scales are arranged so that the coal passers must wheel all coal used from the bins over the scales before they can reach the testing floor. As a further check on the coal consumed during each test, wherever hoppers are used with stokers, etc., the magazines are specially made and calibrated.

Because the trend of consumer demand is toward an ever-increasing freedom from manual attention the laboratory is devoting considerable effort to devices and heating-plant arrangements which make it more convenient for the consumer to heat his home with anthracite.

A home which is cold in the morning, or one in which temperatures are irregular through the day, is not desirable, and to eliminate that condition automatic and semiautomatic damper regulators of all types are investigated in a special compartment in which temperatures may be changed quickly, as desired during the test, by means of refrigeration and radiation systems. The study of the control

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equipment ranges from the simplest and most inexpensive chain and pulley arrangement whereby dampers may be regulated from one of the living rooms of the home to the most elaborate and complete automatic controls which require only the winding of a clock once in every eight days. Meanwhile, the house is automatically maintained at whatever temperatures are desired during the day and night.

All types of domestic stokers are being studied. Because the smaller sizes of anthracite are generally most easily adapted to stoker operation and, due to their relatively low price, the large majority of stokers which are received in the laboratory were designed for the utilization of the smaller sizes. That condition imposes a severe penalty upon the anthracite industry today, but I firmly believe that the problem which it presents will, when properly solved, be largely contributary to the success of the anthracite industry in the future. As smaller sizes come into more general use for house heating, and as they are to an ever greater extent used to furnish automatic heat, their value to the consumer will be increased.

Nor is the development of stokers confined to the home-heating field. Several automatic anthracite-burning devices, whose purpose it is to furnish heat for larger buildings, apartment houses, office buildings, etc., are also being tested and further developed. Particular attention is being given in that connection to the utilization of the rice and barley sizes. It is only logical that if an apartment-house owner, for example, is today paying a given price for a fuel which requires expenditure for labor to fire by hand, he will be only too willing to pay at least an equal price for fuel which he may use in a machine which eliminates the wages of firemen.

It is also a fact that the price of rice and barley sizes to the apartment-house owner must be increased very materially before the cost of heat produced with those sizes and an automatic stoker equals the cost of the same heat with liquid fuels. It has only been during the past few years that any market was available for No. 4 buckwheat. That market is now being further developed, both in special industrial processes and in plants where barley was formerly used. As the demand for No. 4 buckwheat increases and as the market value of all the smaller sizes is raised, the price of the larger sizes may be decreased proportionate to the production of the various sizes. That is the aim of the Industry.

In the stokers designed for home heating, both overfeed and underfeed principles of feed are utilzed.

In one popular type of stoker, anthracite of either buckwheat or rice size is shoveled into a hopper holding one to two days' supply; from the hopper it is carried by a screw conveyor to a fire pot, where air under forced draft is admitted to provide combustion. The ashes fall over the side of the fire pot to the ashpit floor, where they are in turn picked up by a second screw carrying them outside the boiler and depositing them in a covered ashcan. Operation of stokers of this type is controlled by thermostats. The only manual attention required is shoveling the coal into the hopper and removing the ashcans from the basement to the curb for collection. One stoker manufacturer has added an escalator mechanically to convey the coal from the bin to the stoker hopper. The ash conveyor is also arranged so that two or more large ashcans may be filled successively without being replaced.

In other instances the boiler and coal bins are placed so that the coal is fed by gravity from the coal bin to the stoker.

Other stoker manufacturers offer a combined boiler and stoker unit for the builders of new homes. In one such unit the smaller sizes of anthracite are admitted in automatically regulated amounts to a revolving grate, the helical shape of which causes the ash, as the coal is burned, to move toward the center of the grate, where a hole is provided through which the ash drops to a can pitted under the boiler. Another boiler and stoker unit is designed so that anthracite is fed from the hopper by gravity to a sloping grate. The grate is held by springs against a revolving cam, shaped like the cam on a Corliss engine. The revolution of the cam is automatically regulated, making an average speed of one revolution in two minutes. The revolving cam places an increased tension on the spring until the cut-off is encountered. When the grate snaps backwards, with a travel of about ⅛ in., fresh coal is admitted to the grate, and the ashes on the lower end of the grate are dropped off to an ashcan in the ashpit. The boiler is of the steel tubular type.

There is another stoker the sole purpose of which is to fire coal. This accomplished by the use of a reciprocating mechanism which pushes the coal from the hopper, through a tube inserted in the feed door, and drops it on the fuel bed. No provision has been made to date to remove the ash mechanically, although the manufacturer advises that he is developing an ash remover to be added where it is desired.

Another coal-feeding stoker without ash remover is designed to spread anthracite on the fuel bed by revolving paddle wheels. It is well adapted to sectional boilers and steel boilers, and is primarily designed for apartment houses where the janitor can clean the fires when required, but can not frequently replenish coal.

Another apartment-house stoker, which to date has been installed in limited numbers, utilizes the chain-grate principle so successfully employed in industrial anthracite-consuming power plants. It is provided with the customary hopper and is supplied with an auxiliary ash-removal device consisting of a horizontal screw conveyor discharging the ash along the floor to an inclined endless chain, which carries it to an ash receptacle.

Another type of stoker for large heating boilers is designed to cause the fuel bed to advance during combustion without undue agitation, by means of a sloping grate with a reciprocating movement of alternate grate bars. This latter stoker has been under investigation and development at the laboratory during the past two months and is rapidly being perfected. Meanwhile, a prominent manufacturer of bituminous stokers has also seen the opportunities for an anthracite stoker to furnish heat for large buildings and, in cooperation with our laboratory, is making very decided progress in the development of another machine employing somewhat the same principles as that last described.

Thermostats and stoker investigation and development are only a part of the work of our laboratory, which also includes further development of service water heaters, standardization of the most efficient firing and boiler operation methods, the determination of relative values of different fuels, research into new uses of anthracite, etc.

When the laboratory investigation reveals imperfections in any device tested, these imperfections and recommendations for their correction are brought to the attention of the manufacturer, and laboratory approval is withheld until the defects have been remedied. Equipment which fulfills all the requirements of the laboratory and is found to perform in accordance with representations which are made for it, is treated in laboratory reports which are distributed to the anthracite operating companies and all retail companies selling anthracite. These reports provide a description of the device approved and pertinent data concerning its performance, retail sales price, adaptation to boiler sizes, and the value of the device to meet competition with other fuels.

The Anthracite Institute Laboratory has been in operation one year. During that time 76 anthracite-burning devices, including stokers, automatic temperature regulators, forced and induced draft blowers, chemical compounds, gas generators, service-water heaters, etc., have been studied. Devices whose performance has been found satisfactory include five stokers, seven thermostats, two mechanical draft systems, four systems of service-water heating, and one space heater. Other devices which have been studied fall in three classifications: First, tests incomplete; second, devices which are being further developed in the laboratory with the cooperation of manufacturers or inventors; and third, devices which failed in test and have been returned to the manufacturer or inventor for possible further development by him.

To the end that the retail coal merchant may intelligently and confidently advise his customers, and further to the end that consumers may purchase anthracite-consuming devices with the confidence that they will secure the heating results which they expect, the Anthracite Institute Laboratory has recently designed a seal of approval. Authority for the use of this seal on devices and in advertising is given to those companies whose equipment has passed the rigid examination of the laboratory. As the Anthracite Institute's seal of approval becomes more generally recognized it will be of advantage not only to the retail coal merchants and consumers but to the manufacturers of properly designed anthracite-burning equipment.

The Anthracite Institute is satisfied with their investment in research. They are gratified with the universal cooperation of equipment manufacturers and with the use to which retail coal merchants and their salesmen are putting the information which is disseminated from the laboratory.

The future contribution of the laboratory to the anthracite industry in maintaining and increasing the market for anthracite and in providing the consumers with the ultimate in economical and safe heating comfort is certain to be of increasingly great magnitude.

Results of Present
Anthracite Roll Practice

By Paul Sterling*

IN THESE surroundings, near the geographical center of the Eastern bituminous coal field, it may seem presumptuous to introduce anthracite. I do not, however, hesitate to do so, for the fact that for many years a discriminating public in these parts of the United States demanded it and would accept no substitute.

The early history of anthracite and its association with our state governments, who not only gave authority to railroads to construct and maintain roads to transport anthracite coal, but also lent the credit of the state to provide funds to carry out the project, as the State of New York did in 1826 for the Delaware & Hudson Coal Company. In 1830, the Legislature of Pennsylvania granted a charter to the Beaver Meadow Railroad and Coal Company to build and maintain a road to transport coal, and again in 1836 to the Hazleton Coal Company (Lehigh Valley Railroad) to mine, transport and sell coal. The D., L. & W. Railroad was, by its charter, authorized to acquire and hold coal lands and to mine, purchase and sell coal, as well as to transport it.

The railroads had been a great influence in the economic development of anthracite but, in spite of the fact, in 1915, the entire structure of railroad ownership in, or control of coal mining operations, was attacked by the Federal Government, and, in 1921, a decree was entered ordering complete segregation. This building of a great economic industry, by state cooperation and support, which was suddenly destroyed by Federal Government attack, caused, in my opinion, a certain lack of confidence in anthracite, and has contributed to a certain extent to loss of business, or, perhaps, a change in demand for sizes, especially the three largest all-profit-bearing ones. Consider this factor, which for a period exerted an important influence and entirely disappeared.

Lump coal for blast furnaces came into use about 1840, and increased rapidly until 1855, when 380,000 tons were used. In 1890, a maximum of 2,500,000 tons were used, decreasing to 727,000 in 1910, 65,000 in 1920, and not a ton in 1928. During this period "steamboat and broken" sizes were the chief steam fuels for railroads and steamboats. About 1870, anthracite found favor in the manufacture of water gas, and continued until World War conditions, with their attendant high labor costs, necessarily advanced prices, and gas companies discontinued its use, and a further decline in production ensued.

Anthracite production is divided into the following sizes: Lump, steamboat, and broken, known as industrial coal; egg, stove, nut and pea, the domestic sizes; buck, rice and barley, the steam sizes.

All sizes, lump to nut, inclusive, were sold at a profit; not so with pea and smaller. With pea coal selling at $4.50, buck at $3.00, rice at $2.00, and barley at $1.50, the loss on every ton sold is between $1.50 and $4.50 per ton. The Bureau of Mines reports in 1928 the average realization on all steam sizes as $2.12, or an average loss of about $4.00, or approximately $72,000,000, which had to be borne by the domestic sizes.

* Mechanical Engineer, The Lehigh Valley Coal Co.

TABLE I. SHOWING CHANGE IN PERCENT OF SIZES MADE

Year	Industrial	Domestic	Total	Pea	Steam	Total
1890 ...	26	52	78	12	10	22
1926 ...	2	65	67	5	28	33
1928 ...	..	61	61	9	30	39

In 1890, the production of "industrial sizes" constituted 26 percent of the total, and steam sizes 10 percent. In 1928, there were no industrial sizes made, while the steam sizes had increased up to 30 percent of the output. This change was accomplished by breaking down the sizes larger than egg coal in rolls, resulting in a loss of approximately 14 cents per ton into fines.

Operating officials find fault when the breaker refuse contains coal, and complain of the loss. While high recovery is important, it is not, in my opinion, nearly as momentous as the unrecover-

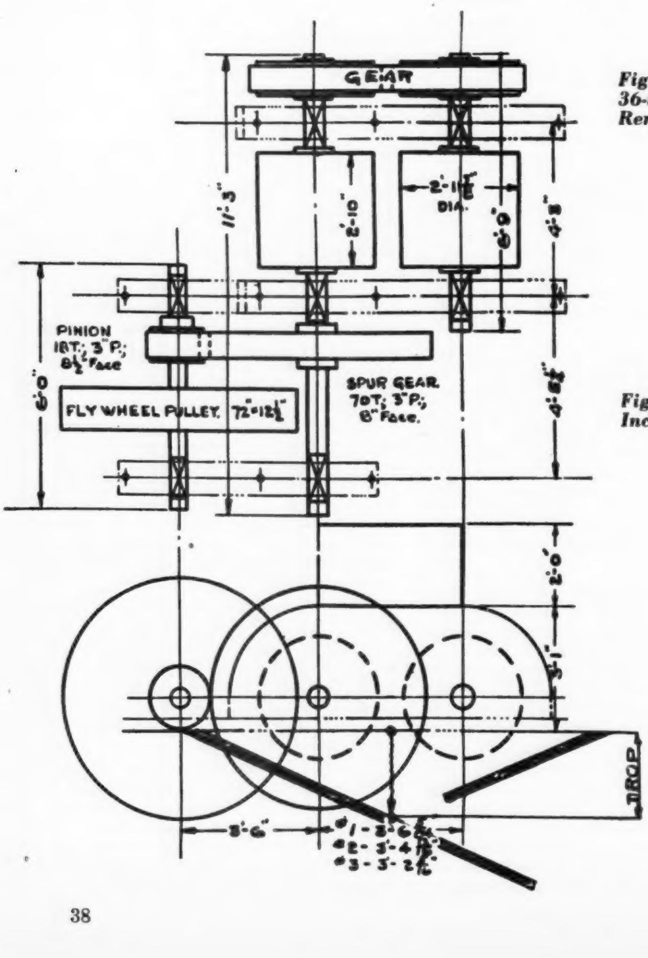


Figure 1. (Left) 36-in. x 34-in. Roll Removable Segments

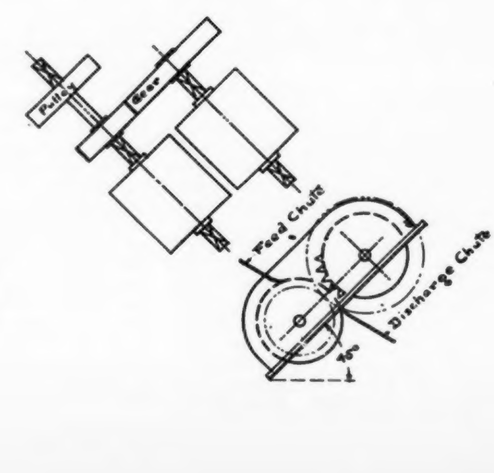


Figure 2. (Below) Inclined Roll—one smooth

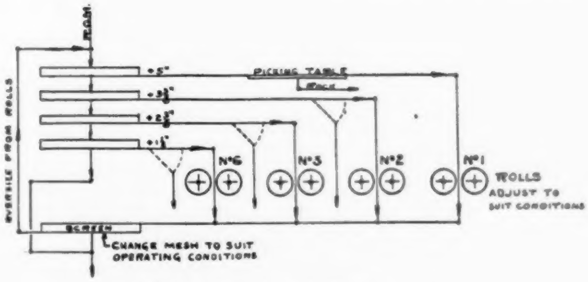
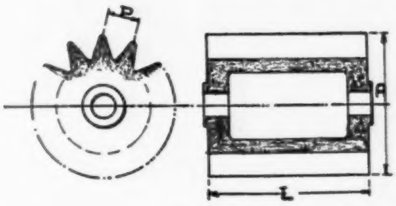


Figure 3. Closed Circuit Grinding



		D	L	P
NUT	22	7[illegible]	12	1[illegible]
NUT	30	11"	16	1[illegible]
STOVE	18	10	24	[illegible]
EGG	18	12	24	2[illegible]
BRO.	20	16	24	2[illegible]
ST. BT	20	26[illegible]	36	4[illegible]
LUMP	13	31[illegible]	36	5[illegible]

Figure 4. Coxe Fluted Roll

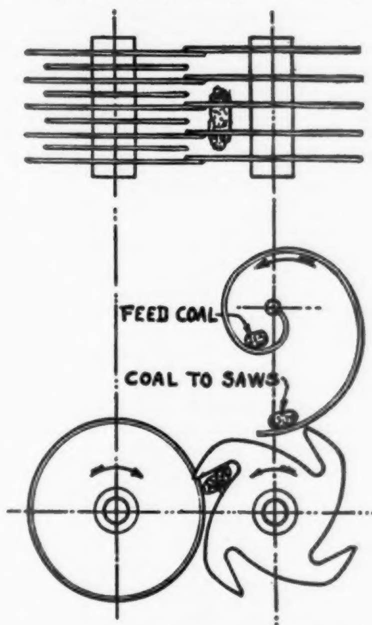


Figure 5. Pardee Gang Saw Roll

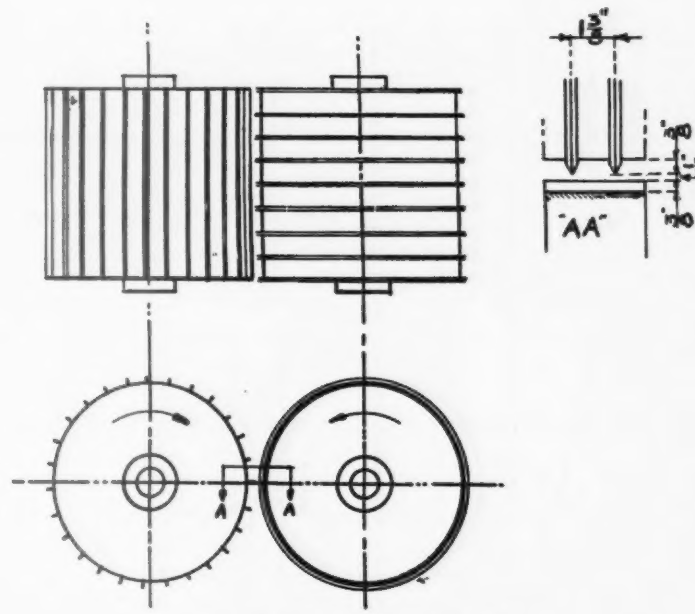


Figure 6. Pardee Flat Roll

able fines made by crushing in rolls. To give you some idea of the loss as compared with high recovery, take 1,000 tons of R. O. M. containing 14.5 percent refuse, or 855 tons of coal and 145 tons of rejects. If the average breaker refuse contains 5 percent coal, the approximate value, at 1930 prices, of the 5 percent coal lost in 145 tons of rejects, is $45. If the 855 tons of coal were shipped in all sizes from broken down to barley, inclusive, the return would be $6,200. However, if the broken and egg sizes were broken down to stove and smaller, in the average roll, the value of the resulting product would be $6,075, or a loss due to degradation of $125, or nearly three times the saving due to 100 percent recovery. Up to 1840, all coal had been shipped as mine-run, and it was a common sight, in Philadelphia, to see the householder breaking the coal on the street. In 1849, the operators began to break the coal at the mines, and screen out the pieces too small to use in the heater. The coal was broken with sledges on perforated plates. In 1845, machinery was introduced for breaking the coal, and as this did the work of 40 to 50 men, it was an economic success.

The introduction to my subject gives: *First,* The causes requiring the introduction of rolls for size reduction; *Second,* Some idea of the monetary loss due present-day practice.

Identification of anthracite rolls is by name or number, i. e., No. 1 on crusher; No. 2 and No. 3 or rebreaker; No. 4, No. 5 and No. 6 or Bony. The use of numbers for rolls was probably due to the fact that anthracite coal sizes were numbered—Steamboat No. 1, broken No. 2, egg No. 3, stove No. 4, nut No. 5 and pea No. 6—and the roll number indicated the size into which the feed coal, to the roll, was broken, i. e., No. 1 roll broke lump coal into No. 1 size or steamboat; No. 2 roll, steamboat into No. 2 or broken, etc.

Such terms as driven tooth roll, segment roll, corrugated roll, fluted roll, etc., all refer to mechanical design rather than type of service. As to results of roll crushing, the operator is chiefly concerned, with the percentage of "industrial" and "domestic" sizes made, above pea coal, as those sizes are the only ones, today, sold above production cost. Roll efficiency may be defined as, "The total percentage of all sizes made above pea coal, from the size fed to the roll." (This definition of efficiency will be used in this article.)

The proper diameter of the roll body, to be used, for the various sizes of feed coal is a much debated subject. In my opinion, roll efficiency depends little on the roll diameter. If smooth rolls were used, where the "angle of nip" varied for the different sizes of feed, the diameter of the roll body would be important. But when toothed rolls are used and the feed coal is drawn in by the points of the teeth, the diameter. within reasonable limits, cuts little ..gure. I have tested practically all diameter rolls from 18 in. up to 54 in. diameter, and have come to the conclusion that a 36-in. diameter roll body, for all sizes of feed coal, will give about as high average efficiency as might be obtained by using various roll diameters for different size coals, i. e., 54 in. for No. 1, 48 in. for No. 2, 42 in. for No. 3, 36 in. for No. 4, etc.

Standardization of roll sizes has the outstanding advantage of standard parts, and the ability to quickly change the roll teeth (if a segment roll) to meet rapid changing market demands.

Large diameter rolls for breaking unusually big lumps may have some advantage, especially when there is a great quantity, as is often the case with stripping coal, but usually the quantity of large pieces is limited, and the oversize lump can easily be broken by hand on the picking table, to feed size, without delaying the operation.

The peripheral speed of the present type of rolls, has an important effect on its efficiency. Tests indicate that 250 ft. per minute is a reasonably efficient one as compared to 900 ft. per minute, generally used before 1910. Breakage,

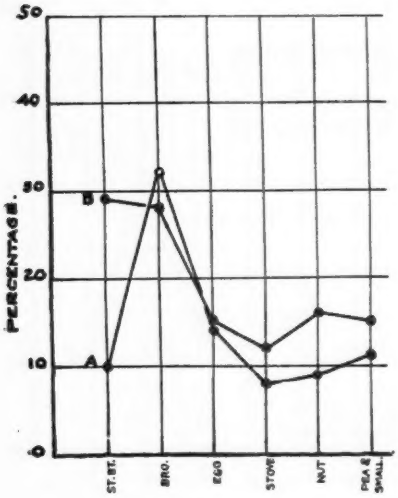


Figure 7

Rolls—34" x 36" at 25 r.p.m. Teeth—Hawkbill, 4" high, 5¼" P. Feed—Lump coal, over 6½". A—Roll centers, 42⅛". B—Roll centers, 42⅜".

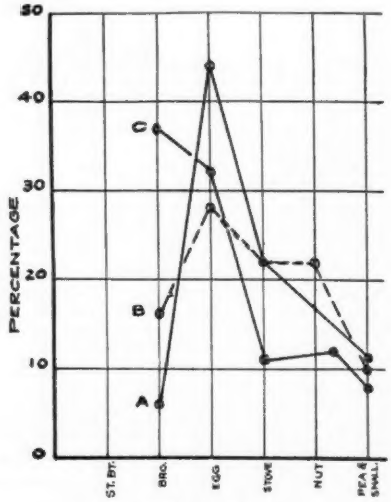


Figure 8

Rolls—34" x 36" at 25 r.p.m. Teeth—Pyramid pointed, 2½" x 1⅜", 3¾" P. Feed—St. Boat, through 6½", over 4½". A—Roll centers, 40¾". B—Roll centers, 40⅞". C—Roll centers, 41⅛".

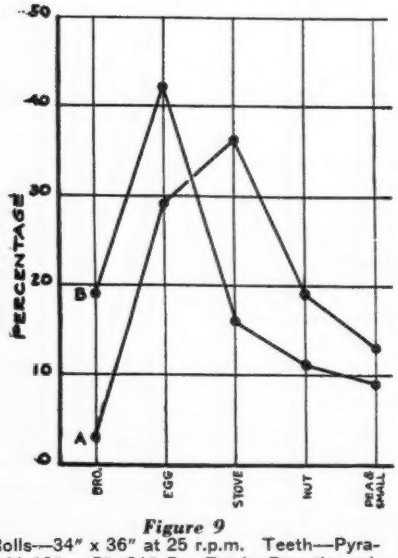


Figure 9

Rolls—34" x 36" at 25 r.p.m. Teeth—Pyramid 1⅜" x ⅞", 2¼" P. Feed—Bro. through 4½" over 3¼". A—Roll centers, 38⅜". B—Roll centers, 38⅝".

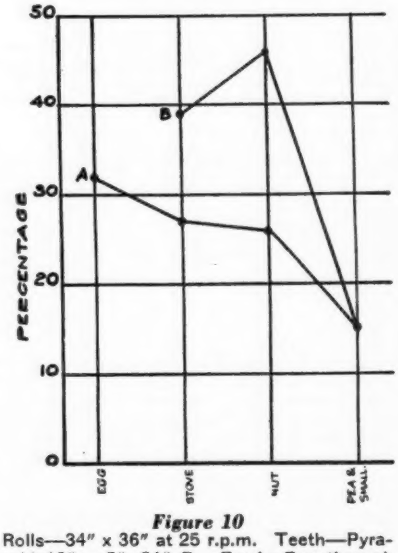


Figure 10

Rolls—34" x 36" at 25 r.p.m. Teeth—Pyramid 1⅜" x ⅞", 2¼" P. Feed—Egg through 3¼", over 2 5/16". A—Roll centers, 38⅜". B—Roll centers, 38⅝".

due to speed, is caused when the coal released from the roll, falls on the chute underneath it. The height of fall or head, is equal to the velocity head imparted to the coal by the roll plus the static head (usually about 2 ft.).

If the breakage, due to the drop under the roll, is eliminated, a saving in fine degradation may be expected. Little has been done to overcome or reduce the breakage due to this cause. The writer has designed and tested a set of rolls with this in view, and shop tests indicated an efficiency of 93 percent. They consisted of a smooth and a toothed, geared in the standard way, and set on an incline of 45 degrees. The smooth roll below. Coal is fed onto the smooth roll, back of its vertical centre, and acts as a feeder drawing the feed coal into the rolls. The smooth roll is an anvil on which the coal is broken by the teeth on the upper roll. The discharge from the rolls slides onto a plate, resting on the face of the smooth roll, without drop. The plan is to develop the machine, with concentric ridges on the smooth roll, working with a feeder which will lay each piece of coal with its long axis across the ridges. The tooth on the opposite roll will track between the concentric ridges and strike the coal at its centre. I am satisfied, if this can be accomplished, it will be a step in the right direction.

Laboratory tests, made by hand, indicate an efficiency approaching 100 percent when breaking coal, supported at its ends, and striking it midway between the supports.

I am thoroughly convinced that the most efficient results are obtained when each size is broken to the next smaller, or set of rolls for each size. This plan was not generally in vogue, largely due to the fact that first cost did not warrant the machinery and structure required for the operation. The practice was No. 1 roll breaking to Broken, and No. 2 or No. 3 breaking Broken to Egg.

Recent changes in market conditions suggest a decided change, and I believe a closed circuit grinding installation will be the future plan. Such a system will permit an oversize being made in each roll, which tends to maximum efficiency, and returning this oversize to be reground. (*See Figure 3.*)

The "Driven Tooth" roll consisted of a cast iron drum into which dropped forged teeth with turned shank were driven into drilled holes in the drum. This design was generally used until about 1910. Since then the "Segment Roll" has come into use—it consists of a many sided (usually 11) drum, to which are bolted cast iron, alloy steel or manganese steel segments. This design permits standardization of the drums, frame housing, etc., and by changing segments it may be adapted to varying feed sizes. One detail, a relic of early days, requiring consideration is the gears. They are designed with unusually long addendum teeth, permitting a small range of adjustment, when the roll centers are greater or less than that for which the gears were designed, they are changed for a larger or smaller pitch diameter gear. This arrangement prevents a close adjustment. If a train of gears could be used, and designed to permit adjustment from minimum to maximum roll centers, it would be a decided advantage. The total movement would not exceed 6 in.

The use of cushion springs, to protect rolls from foreign material, such as sulphur balls, tramp iron, etc., if too rigid, are of little protection, and if too soft, allow excessive oversize to be made, especially when grinding very hard anthracite. The use of cast iron breaking shells for protection is fairly reliable, without the objection there is to springs.

Mr. Eckley B. Coxe, a man of great vision probably did as much, if not more, than anyone to reduce the wasteful practices in vogue in the early days of anthracite manufacture. He is responsible for the "Fluted Roll" (*Figure 4*) and to a large extent to the practice of grinding by size. The fluted roll was an advance in roll practice, but the present design of teeth with improved spacing promotes better efficiency than the fluted roll of his time.

Mr. Frank Pardee, Hazleton, Pa., designed an experimental roll which gave

TABLE II. ESTIMATED ROLL BREAKAGE DUE TO DROP UNDER ROLL.

	Head			Percent Breakage into Smaller Sizes					
				Broken		Egg		Stove	
Speed ft. per min.	Vel. ft.	Static ft.	Total ft.	Above pea	Below pea	Above pea	Below pea	Above pea	Below pea
900	3.5	2	5.5	5.36	6.35	6.36	6.35	3.82	4.04
250	.25	2	2.25	3.98	2.15	4.98	2.15	2.74	2.61

TABLE III. SIZE OF ROLLS, TOOTH SPACING AND SIZE COMMONLY USED IN THE ANTHRACITE FIELD

Size Roll	Number	Number Segments	Number Teeth Circle	Number Teeth Row	Number Teeth C. to C.	Tooth High, In.	Tooth Sq. In.		Coal Feed	Coal Discharge	
51 x 41	1	Solid	36	8	4½	3½	1¾		Lump	St. Boat & Bro.	A
36 x 34	1	11	11	6	5¼	4		Hawkbill	Lump	St. Boat & Bro.	B
			11	6	5¼	3		Chisel Point			
30½ x 48	1	4	11	6	5¼	1½		Chisel Point			
			8	5	8⅞	5¼	4	Staggered	Large	St. Boat & Bro.	C
33½ x 46	1	Solid	8	4	8⅞	5¼	4		Lump		
			19	8	5½	3½	1¾		Lump	St. Boat & Bro.	D
36" x 34	2	11	22	10	3	2 5/16	1½	Staggered	St. Boat	Broken	E
			22	9	3	2 5/16	1½				
36 x 34	2	11	44	10	2⅞	2 5/16	1½		St. Boat	Broken	F
36 x 34	3	11	55	13	2 3/16	1⅝	1¼		Broken	Egg	G
16¾ x 21	4	8	40	13	1¼	1¼	⅞		Egg	Stove	H
18 x 30	4	4	20	12	2¼	2	1¼	Staggered	Egg	Stove	I
			20	11	2¼	2	1¼				
16¾ x 21	6	8	.2	22	¾	¾	¾		Stove & Nut	Pea	J
30 x 36	2	Solid	26	10	3⅜	2¾	1½		St. Boat	Broken	K
30 x 36	3	Solid	39	14	2¼	2	1¼		Broken	Egg	L
24 x 36	3	Solid	31	14	2¼	2	1¼		Broken	Egg	M
21½ x 30	3½	Solid	29	12	2.33	2	1¼				N
21½ x 30	4	Solid	54	17	1.64	1¼	⅞		Egg	Stove	O
24 x 30	6	Solid	76	21	1.35	1⅛	1		Stove & Nut	Pea	P

TABLE IV. THE FOLLOWING AVERAGE RESULTS FROM ROLLS, AS PER TABLE III, INDICATE THE BENEFITS OF SLOW SPEED.

Size Roll	Style	Speed Ft./Min.	Feed Coal	Steam Boat	Broken	Egg	Stove	Nut	Eff.	Table 3		
36 x 34	1	250	Lump	27	23	22	10	9.0	91 %	A		
36 x 34	1	942	Lump	29	24	13	8.9	7.7	82.6%	A		
30½ x 48	1	...	Lump	76.6	9.0	5.0	3.8	3.0	97.4%	C		
36 x 34	2	230	Steam Boat	....	38.8	36.3	12.7	5.6	93.4%	F		
30 x 36	2	900	....	....	29.0	30.0	15.0	11.0	85.0%	K		
36 x 34	3	250	Broken	....	7.0	25.0	40.0	17.0	89.0%	G		
30 x 36	3	900	Broken	....	8.0	23.0	30.0	13.5	74.5%	L		
35 x 34	3	250	Egg	....	....	45.	23.	17.0	85.0%	G		
30 x 36	3	900	Egg	....	....	17.5	41.	21.5	80.0%	L		
36 x 34	3	250	Stove	....	....	....	45.5	43.	88.5%	G		
36 x 34	3	...	Steam Boat	....	13.7	43.7	18.8	12.6	88.8%	E		
36 x 34	6	250	Stove	....	....	....	28.9	54.2	83.1%			
16¾ x 21	6	530	Flat Stove	....	....	....	4.0	62.	......	Pea	Buck	J
16¾ x 21	6	530	Flat Nut	....	....	....	....	33.8	......	34.4	13.4	J
16¾ x 21	6	530	Flat Stove	....	....	....	....	67.1	......	11.4	8.6	H
16¾ x 21	6	530	⅞" Nut	....	....	....	....	26.5	......	28.5	9.5	H

TABLE V. RESULTS GRINDING LUMP AND STEAMBOAT AT VARYING ROLL CENTERS.

Roll	Centers	Speed	Feed	Steam Boat	Broken	Egg	Stove	Nut	% Eff.	
36 x 34	42¾"	250	Lump	25.	33.	13.0	9.5	9.0	89.5	B
	42⅞"	250	Lump	27.5	22.5	18.5	11.5	11.5	91.5	B
	43"	250	Lump	30.0	24.	12.	14.	10.	90.	B
	43⅛"	250	Lump	31.0	21.0	13.5	13.	11.	89.5	B
	43¼"	250	Lump	45.0	13.5	14.5	9.5	7.5	90.0	B
	43⅜"	250	Lump	45.5	16.0	13.0	8.5	8.0	91.0	B
	39¼"	250	Broken	....	....	15.	45.5	21.5	82.	G
	39½"	250	Broken	....	....	21.5	40.0	20.5	82.	G
	39⅝"	250	Broken	....	....	29.	36.0	19.5	84.5	G
	39¾"	250	Broken	....	....	24.5	42.	19.0	85.5	G
	39⅞"	250	Broken	....	....	33.5	37.5	15.5	86.5	G
	40.0"	250	Broken	....	....	39.5	30.5	17.5	87.5	G
	40⅛"	250	Broken	....	....	48.0	24.5	14.5	87.	G
	40¼"	250	Broken	....	....	60.0	17.5	12.0	89.5	G

TABLE VI. GRINDING EGG FLATS AND STD. EGG IN PARDEE FLAT ROLLS.

Feed	Stove	Nut	Pea	Buck	Rice	Barley	Smaller
....	2.4	67.	9.5	8.4	5.5	3.6	3.6
....	6.	71.	7.7	5.7	4.	4.	1.6

TABLE VII. NORTON VERTICAL PICK VS. HUDSON 3-ROLL SERIES GRINDING.

Feed	Norton Breaker Percent Prep. Sizes	Norton Breaker Percent Steam	Hudson Coal 3 Percent Prep. Sizes	Roll-Series 1, 2 and 3 Percent Steam	Fine
12" to 14" Lump	87.02	12.98	84.3	14.5	1.2
6" to 11" Lump	85.80	14.20	86.3	13.	0.7

approximately 96 percent efficiency (*Figure 5*). It consists of several steel plates, circular saws, of special design, mounted on one shaft and opposite to them on a second shaft an equal number of steel discs. The saws and discs revolve towards each other, coal is fed on top so that its long axis is parallel to the shafts. The distance between the saws being equal to the ring diameter through which the coal is to pass. (*Figure 5.*)

A later design installed by the Lehigh Navigation Co., *Figure* 6, was experimented with grinding flat coal, with the results shown in Table VI.

I am of the opinion that continued experiments, with varying spacing of cutters, may be productive of a design, for flat fracture coal, objected to by the customer, that will suggest their use for manufacturing pea and buck sizes.

The Norton Limited, Tipton, Staffordshire, England, conducted tests for the Hudson Coal Company, on Pennsylvania anthracite, with their Norton Vertical Pick Breaker. The machine consists of a movable platform on which the coal is fed and moved forward, under the vertical picks, which descend breaking the coal. The platform and picks are synchronized so that on the down stroke the platform is stationary—on the up stroke the platform moves forward in fixed increments. The picks are placed in series on a walking beam and so arranged that the first picks with the longest stroke break the large lumps, and the shorter stroke ones do progressive breaking as the platform advances. The machine seems to have merit but further development and tests are necessary before conclusive evidence can establish its superiority to the present rolls.

Test of grinding in gyrators, jaw crushers, single roll tooth cushions, smooth rolls, ring crushers, pulverizers, stamp mills, have not shown to date as high an efficiency as the toothed rolls in use. So there is not, in my opinion, any reason to change from the present machine.

I do know that sharp pointed teeth and slow periphery speed have a decided influence on the efficiency. At least monthly inspection and test should be made of all rolls, in order that the machine may be maintained in as near perfect condition as possible. When you realize that 1 percent difference in efficiency is approximately 8 cents per ton on the cost, then you will see the importance of constant negligence.

With the decreasing demand for domestic sizes, the operators are greatly concerned with roll efficiency. There is very little information on the subject, and they certainly welcome any constructive information that will be beneficial to the industry.

Figures 7, 8, 9 *and* 10, show results from Nos. 1, 2 and 3 rolls, when breaking lump to steamboat, steamboat to broken, broken to egg and egg to stove, respectively, varying the amount of oversize made by increasing the roll centers. These results are the average of several thousand tests, and fairly represent the average present day practice.

PREPARATION of ANTHRACITE FINES

By E. P. Humphrey*

ANTHRACITE fines is that portion of the anthracite product smaller than barley coal. Considerable barley coal is now being made over 3/32-in. round mesh, and so for the subject under discussion all the material passing 3/32-in. is classed as fines. These fines are known by various other names, as slush, silt, culm, dust coal, anthrafines or No. 4 and No. 5 buckwheat.

The amount of the fines, varying with the regions, is from 8 to 25 percent of the colliery shipments. The foregoing percentage includes fines in the overflow from settling tanks, generally not considered. The collieries in the Southern Field, due to the character of mining and greater friability of their coal, have more fines than any other field, the Northern Field the least.

The fines in their original unprepared state are a mixture, therefore, of fine particles of coal, slate, bone, rock or sand and clay in varying proportions as to size and quantities.

It is estimated for the anthracite industry there is 10,000,000 tons of fines per year available for recovery. Some companies make no recovery, other intermittently. One company of its available run of mine fines of 450,000 tons for the year shipped to market 54,000 tons, or 12 percent. The prepared product was of various ash contents and different sizing specifications. The fines are generally present in the breaker wash water in different proportions up to 20 percent by weight.

The amount of water required to prepare anthracite in a breaker, considering all sizes, is considerable. This washing water is finally collected at the end of the process and contains all the fine coal, in addition to any other fine foreign material of that size present in coal when brought from the mines. A 1,000-ton plant in 8 hours requires in many cases from 2,000 to 3,000 gallons per minute, whereas a 5,000-ton plant normally uses 7,000 gallons per minute. Often in plants of this size, wash water is re-circulated after removal of fines. The dirtier and muddier run of mine, with an increased percent of fines, requires a lot more water than the colliery having cleaner hand loaded run of mine. Many operations impound the wash water in dams to settle out the fines simply to reduce stream pollution, making at the present moment no recovery of the fines for market. Some have settling plants arranged to recover a major portion of the fines, part or all of which is prepared for market, the remainder and refuse wasted. Others, on account of water shortage, find it necessary to clarify the water for reuse; the fines then may either be used or wasted.

The treatment of the fines may be divided into two processes—first, the separation of them from the water, and second, the preparation of the product as to sizing and ash reduction.

SEPARATION

The separation for further treatment is generally accomplished by settling out the fines in large tanks, the size increasing with the amount of water and included fines to be handled, and clarity of overflow desired. Some are long horizontal tanks with a conveyor running slowly along the bottom, others are of the deeper type with submerged elevators, and there is the circular tank manufactured by the Dorr Company, called a thickener, with the bottom discharge of the settled material. Each of these types can be designed to make efficient separation.

METHODS OF SETTLING

Settling Tank—Lehigh Navigation Coal Co. Alliance Colliery

A rectangular tank 98 ft. long, 10 ft. wide, and 9½ ft. deep inside, with a double-strand conveyor with 6½-ft. long flights, running at 42 ft. per minute; return line above water along top, treating 2,300 gallons per minute incoming water and recovering 35 tons per hour of run of mine fines. The cubical capacity of this tank is 64,000 gallons; the inflow 2,300 gallons per minute, containing 20.8 percent solids, overflow 2,100 gallons per minute with 6.67 percent solids; recovered fines contain 55 percent water. The reason for the large amount of fines in the overflow is due to the amount of minus-200-mesh material in circulating water. This accumulation is due to reusing considerable of the overflow again and again in the breaker. (See Table I.)

DORR HYDROSEPARATOR—*Lehigh Navigation Coal Co. Greenwood Colliery*

The original size of this machine, 16 ft. 6 in. diameter, was reduced to 14 ft. 6 in. diameter to shorten settling time. This decreased the amount of fines in the recovered underflow.

The cubical content of this tank is now 8,000 gallons. The inflow is 3,000 gallons per minute containing 8 percent solids and the overflow is 2,500 gallons per minute containing 2.25 percent solids. This separator recovered 40.8 tons of solids in underflow per hour containing 73 percent water. (See Table II.)

At the Loomis Colliery of the Glen Alden Coal Company there is in operation an 85-ft. diameter Dorr thickener. This outfit receives all the fines and water from the breaker through a 3/64-in. mesh screen, the plus 3/64-in. material being screened out in the

TABLE I

		Recovered fines Percent size	Recovered fines Tons in 8 hours	Overflow Percent size	Overflow Tons in 8 hours	Inflow Tons in 8 hours	Inflow Percent size
	On 8 Mesh	2.25	6.3	......	.....	6.3	1.21
Through 8	" 14 "	17.40	48.7	......	.....	48.7	9.18
" 14	" 20 "	13.9	38.9	......	.....	38.9	7.36
" 20	" 28 "	16.2	45.4	......	.....	45.4	8.58
" 28	" 35 "	14.6	40.8	......	.....	40.8	7.71
" 35	" 48 "	10.9	30.5	......	.....	30.5	5.75
" 48	" 65 "	8.7	24.4	1.5	3.8	28.2	5.32
" 65	" 100 "	7.3	20.5	2.0	5.0	25.5	4.81
" 100	" 150 "	3.5	9.8	6.0	15.0	24.8	4.68
" 150	" 200 "	1.75	4.9	2.0	5.0	9.9	1.88
" 200	"	3.50	9.8	88.5	221.2	231.0	43.52
		100.00	280.0	100.00	250.0	530.0	100.00

TABLE II

		Underflow Percent size	Underflow Tons in 8 hours	Overflow Percent size	Overflow Tons in 8 hours	Inflow Tons in 8 hours	Percent size
	On 8 Mesh	.4	1.3	......	.....	1.3	.31
Through 8	" 14 "	15.8	51.6	......	.....	51.6	12.15
" 14	" 20 "	15.2	49.6	.5	.5	50.1	11.80
" 20	" 28 "	17.4	56.8	.5	.5	57.3	13.46
" 28	" 35 "	15.8	51.6	2.0	2	53.6	12.58
" 35	" 48 "	11.2	36.5	3.0	3	39.5	9.26
" 48	" 65 "	8.1	26.4	5.5	5.5	31.9	7.47
" 65	" 100 "	7.1	23.1	16.9	16.9	40.0	9.40
" 100	" 150 "	3.3	10.7	14.4	14.4	25.1	5.85
" 150	" 200 "	1.2	3.9	8.5	8.5	12.4	2.87
" 200	"	4.5	14.5	48.7	48.7	63.2	14.85
		100.00	326.0	100.00	100.0	426.0	100.00

* Supervisor of Preparation, Lehigh Navigation Coal Company.

breaker. Added to this at present is the re-wash water and fines from Rheolaveur plant working on the underflow from this thickener. It is estimated the inflow is 6,400 gallons per minute; the overflow at 5,000 gallons per minute and underflow with solids pumped to Rheolaveur plant of 1,400 gallons per minute.

The percent solids in overflow is 0.403 and would probably be less if re-wash from Rheolaveur plant was not returned. This plant at time of test had been in operation only a few weeks, and it is later expected to keep re-wash in circulation at Rheo system.

THICKENER OVERFLOW

	Mesh	Percent Size	Percent Ash	Cumulative Size	Cumulative Ash
Thru 150	On 200	1.4	51.60	1.4	51.60
Thru 200		98.6	47.98	100.0	48.01

The sample of underflow which is pumped to Rheolaveur plant in breaker is not a representative sample of the feed as it would be if re-wash was not returned to the thickener and mixed with regular feed. It is believed the regular feed would be slightly lower in ash. The re-wash corresponds to middlings in another operation, and is not rejected but returned to system for further treatment.

The underflow is pumped to a four-rake unit Dorr classifier at the head of the Rheolaveur plant to remove excess water from fines. The overflow from this classifier is 1,200 gallons per minute, containing 3.81 percent solids. The rake product from the classifier going direct to the Rheo plant as feed, the overflow of classifier to waste.

CLASSIFIER RAKE PRODUCT RHEO FEED, INCLUDING RE-WASH MATERIAL

		Mesh		Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
		On	8	....	.....	.....	.....
Thru	8	"	14	4.2	11.27	4.2	11.27
"	14	"	20	4.2	14.98	8.4	13.12
"	20	"	28	24.3	16.24	32.7	15.44
"	28	"	35	31.5	20.61	64.2	18.05
"	35	"	48	16.4	27.41	80.6	19.97
"	48	"	65	9.0	47.52	89.6	22.80
"	65	"	100	6.0	51.78	95.6	24.59
"	100	"	150	2.6	52.36	98.2	25.34
"	150	"	200	0.6	50.91	98.8	25.39
"	200			1.2	45.63	100.0	25.69

CLASSIFIER OVERFLOW TO WASTE

		Mesh		Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
		On	20	....	.....	....	.....
Thru	20	"	28	1.2	8.22	1.2	8.22
"	28	"	35	5.6	7.63	6.8	7.74
"	35	"	48	8.7	9.33	15.5	8.62
"	48	"	65	10.7	12.65	26.2	10.27
"	65	"	100	11.9	21.94	38.1	13.92
"	100	"	150	6.9	21.43	45.0	15.10
"	150	"	200	1.8	24.94	46.8	15.50
"	200			53.2	48.37	100.0	33.18

PREPARATION

To get some conception of the relation of the different sizes of the particles in a sample of raw fines, consider the largest particle or that which has the cross section of 3/32 in., or about No. 8 mesh, and the size of a particle of 100-mesh. In many slush samples there is considerable material much smaller, some through 200 mesh.

The cross sectional area of the larger or 3/32-in. sized piece is 258 times greater than the cross section of the pieces the size 100 mesh. Carrying the comparison to one that may be better visualized, the same relation as to cross section area exists between a piece of small broken coal, 3½ in. in diameter, and a piece of buckwheat ¼ in. in diameter. Is it any wonder that this mixture of fine material presents such a problem in cleaning, with its range of sizes and gravities?

There is at present no general standard specification for the prepared fine product. Some consumers want it raw with no restriction as to ash or sizing; this, then, can be loaded from a settling tank, in the case the fines do not carry much more than 50 percent water, direct into railroad car, the excess water filtering out the doors of the car. In case of loading with excess water, as is generally experienced with underflow from Dorr thickener, a Dorr classifier is used to make necessary moisture reduction.

There are a great number of possible specifications as to ash and sizing, and the Lehigh Navigation Coal Company is now making fines in the following classifications:

Class A Material through 3/32-in. or 8 Mesh, 10 to 15 percent Ash, no size restriction.

Class B Material through 3/32-in. or 8 Mesh, 15 percent Ash and over, no size restriction.

Class C Material through 3/32-in. over 1/32-in. or 28 Mesh, 10 to 20 percent Ash, undersize 20 to 25 percent.

Class D Material through 3/32-in. over 3/64-in. or 20 Mesh, 10 to 20 percent Ash, undersize 20 to 25 percent.

Class E Material through 3/64-in., 10 to 20 percent Ash, no size restriction.

The specification required determines largely the method; when there is a sizing restriction along with ash reduction, additional sizing shakers are required. A review of the principal methods of ash reduction that have general use in the field follow; tests show the effectiveness and efficiency of each.

Plato Table

Installation at Greenwood Colliery, Lehigh Navigation Coal Co., consists of five Plato double-deck tables taking underflow without further water reduction from 14½-ft. diameter Dorr thickener, as described above. The performance per table was 8.14 tons per hour feed; to refuse, 4.86 tons per hour and cleaned coal 3.28 tons per hour, or a yield of cleaned coal to feed of 40.2 percent. The coal ends were collected and run to a classifier for water reduction before loading. The slate ends to another classifier for the same purpose before going to slate bank. The additional loss of fines of 2.1 tons per hour of combined cleaned coal was had in coal classifier overflow. The loss of these fines reducing the ash from 11.32 as discharged from table to 10.40 as in car. The overflow in classifier having been purposely increased to reduce ash. An installation at Lansford Colliery of same company has 10 Deister tables set up in practically the same layout, giving the same results.

The details of the sizing and ash of preparation follow:

FEED TO TABLES

		Mesh		Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
		On	8	0.4	15.63	0.4	15.63
Thru	8	"	14	15.8	20.39	16.2	20.28
"	14	"	20	15.2	17.15	31.4	18.75
"	20	"	28	17.4	19.35	48.8	19.00
"	28	"	35	15.8	21.46	64.6	19.57
"	35	"	48	11.2	24.30	75.8	20.28
"	48	"	65	8.1	33.29	83.9	21.53
"	65	"	100	7.1	33.52	91.0	22.45
"	100	"	150	3.3	35.18	94.3	22.92
"	150	"	200	1.2	33.23	95.5	23.08
"	200			4.5	36.91	100.0	23.68

COAL FROM TABLES

		Mesh		Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
		On	8	0.2	6.75	0.2	6.75
Thru	8	"	14	20.7	7.91	20.9	7.90
"	14	"	20	19.2	9.31	40.1	8.58
"	20	"	28	20.5	10.21	60.6	9.13
"	28	"	35	16.4	12.18	77.0	9.78
"	35	"	48	9.4	12.79	86.4	9.96
"	48	"	65	5.2	16.40	91.6	10.32
"	65	"	100	4.2	18.27	95.8	10.65
"	100	"	150	1.9	21.03	97.7	10.88
"	150	"	200	4.4	24.3	98.1	10.99
"	200			1.9	32.40	100.0	11.32

SLATE FROM TABLES

		Mesh		Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
		On	8	0.4	53.74	0.4	53.74
Thru	8	"	14	12.2	40.38	12.6	40.80
"	14	"	20	11.2	48.54	23.8	44.90
"	20	"	28	16.6	38.20	40.4	42.75
"	28	"	35	17.2	38.50	57.6	40.90
"	35	"	48	14.0	46.39	71.6	42.00
"	48	"	65	10.8	37.99	82.4	41.51
"	65	"	100	9.5	46.46	91.9	41.95
"	100	"	150	4.1	58.33	96.0	42.55
"	150	"	200	1.5	56.50	97.5	42.98
"	200			2.5	51.60	100.0	43.12

CLASSIFIER OVERFLOW

		Mesh		Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
		On	8	....	.....	....	.....
Thru	8	"	14	2.7	8.76	2.7	8.76
"	14	"	20	4.6	8.84	7.3	8.81
"	20	"	28	8.9	9.37	16.2	9.12
"	28	"	35	14.3	10.64	30.5	9.85
"	35	"	48	16.3	11.23	46.8	10.33
"	48	"	65	15.5	12.02	62.3	11.45
"	65	"	100	17.1	17.22	79.4	12.69
"	100	"	150	8.9	29.59	88.3	14.39
"	150	"	200	4.3	43.47	92.6	15.72
"	200			7.4	49.28	100.0	18.18

Brookside Colliery—Philadelphia & Reading Coal & Iron Co.

At this plant unsized fines from elevator type settling tank are treated in a hydrotator for ash reduction. The sizing of samples from feed, washed coal and refuse follow:

					Sample Feed		Cleaned Coal		Refuse	
					Size percent	Cumulative	Size percent	Cumulative	Size percent	Cumulative
		On	8	Mesh	1.7	1.7	1.5	1.5	1.8	1.8
Thru	8	"	14	"	19.3	21.0	16.8	18.3	13.2	15.0
"	14	"	20	"	19.3	40.3	18.5	36.8	13.9	28.9
"	20	"	28	"	21.6	61.9	21.9	58.7	22.9	51.8
"	28	"	35	"	15.7	77.6	17.2	75.9	20.6	72.4
"	35	"	48	"	9.6	87.2	10.4	86.3	10.8	83:2
"	48	"	65	"	5.2	92.4	6.0	92.3	7.2	90.4
"	65	"	100	"	3.8	96.2	3.8	96.1	6.0	96.4
"	100	"	150	"	1.1	97.3	1.5	97.6	1.8	98.2
"	150	"	200	"	0.5	97.8	0.9	98.5	0.6	98.8
"	200				2.2	100.0	1.5	100.0	1.2	100.0

The average results for a 47-hour operating period show the following results:

Average Composite Sample Feed	23.18
Average Composite Sample Cleaned Coal (ash)	12.80
Average Composite Sample Refuse	48.84
Average Tons per Hour Cleaned Coal	32.9
Average Tons per Hour Refuse	13.30
Average Tons per Hour Feed	46.20

Results of operation of Rheo plant just recently installed at Loomis Colliery follow; later tests have shown that a reduction in the ash in the washed coal has been made lower than shown here. On the two days following date of present test, composite results were:

	Washed Coal	Refuse
April 23	10.28	59.10
April 24	8.68	54.35

demonstrating that a reduction in washed ash results in a reduction in ash of refuse reject. Quantity of washed fines 8.5 tons per hour.

WASHED COAL—RHEOLAVEUR PLANT

Mesh	Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
On 8	....	.....	....	.....
Thru 8 " 14	1.2	6.08	1.2	6.08
" 14 " 20	2.4	7.48	3.6	7.03
" 20 " 28	22.2	8.59	25.8	8.37
" 28 " 35	34.3	12.37	60.1	10.58
" 35 " 48	20.2	18.52	80.3	12.58
" 48 " 65	10.7	23.98	91.0	13.95
" 65 " 100	5.6	33.90	96.6	15.10
" 100 " 150	1.8	35.75	98.4	15.85
" 150 " 200	0.8	35.46	99.2	16.00
" 200	0.8	36.49	100.0	16.12

REFUSE REJECT—RHEOLAVEUR PLANT

Mesh	Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
On 8	1.0	39.13	1.0	39.13
Thru 8 " 14	8.3	45.19	9.3	44.50
" 14 " 20	7.2	64.94	16.5	48.22
" 20 " 28	25.2	67.19	41.7	59.69
" 28 " 35	29.1	74.08	70.8	65.53
" 35 " 48	16.6	78.60	87.4	68.02
" 48 " 65	7.9	80.96	95.3	69.04
" 65 " 100	3.7	77.63	99.0	69.38
" 100 " 150	0.6	70.36	99.6	69.38
" 150 " 200	0.2	61.61	99.8	69.36
" 200	0.2	53.15	100.0	69.33

BREAKER WASH WATER TO THICKENER

Mesh	Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
On 8	0.3	20.75	0.3	20.75
Thru 8 " 14	15.1	17.83	15.4	17.87
" 14 " 20	15.0	18.17	30.4	18.05
" 20 " 28	17.6	20.16	48.0	18.80
" 28 " 35	14.2	20.87	62.2	19.22
" 35 " 48	10.3	24.58	72.5	20.00
" 48 " 65	8.7	33.61	81.2	21.42
" 65 " 100	8.5	39.38	89.7	22.18
" 100 " 150	4.7	37.64	94.4	23.84
" 150 " 200	1.3	44.45	95.7	24.08
" 200	4.3	50.13	100.0	25.28

THICKENER OVERFLOW TO WASTE

Mesh	Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
On 65	1.0	11.12	1.0	11.12
Thru 65 " 100	1.5	11.68	2.5	11.49
" 100 " 150	1.4	11.97	3.9	11.62
" 100	0.2	16.00	4.1	11.82
" 200	95.9	48.78	100.0	48.1

THICKENER UNDERFLOW—TABLE FEED

Mesh	Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
On 8	0.8	31.04	0.8	31.04
Thru 8 " 14	19.5	22.93	20.3	23.22
" 14 " 20	17.9	23.67	38.2	23.50
" 20 " 28	19.5	25.32	57.7	24.10
" 28 " 35	14.7	26.56	72.4	24.63
" 35 " 48	9.4	38.62	81.8	26.28
" 48 " 65	6.9	41.65	88.7	27.38
" 65 " 100	6.1	45.60	94.8	28.60
" 100 " 150	2.8	49.39	97.6	29.15
" 150 " 200	0.6	53.50	98.2	29.15
" 200	1.8	54.79	100.0	29.79

TABLE REFUSE TO WASTE

Mesh	Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
On 8	1.9	72.92	1.9	72.92
Thru 8 " 14	10.1	74.64	12.0	74.40
" 14 " 20	8.0	71.07	20.0	73.0
" 20 " 28	9.7	66.76	29.7	70.9
" 28 " 35	12.2	69.11	41.9	70.6
" 35 " 48	12.9	75.31	54.8	71.1
" 48 " 65	15.2	82.95	70.0	73.5
" 65 " 100	14.2	82.25	84.2	75.0
" 100 " 150	7.2	80.02	91.4	75.4
" 150 " 200	4.0	78.58	95.4	75.65
" 200	4.6	73.77	100.0	75.44

CLASSIFIER OVERFLOW TO WASTE

Mesh	Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
Thru 8 On 14	0.5	23.13	0.5	23.3
" 14 " 20	0.5	19.80	1.0	21.6
" 20 " 28	0.9	11.08	1.9	16.59
" 28 " 35	3.7	10.15	5.6	12.37
" 35 " 48	7.1	8.28	12.7	10.10
" 48 " 65	12.0	8.28	24.7	9.19
" 65 " 100	21.9	9.75	46.6	9.49
" 100 " 150	16.7	13.58	63.3	10.58
" 150 " 200	28.7	18.75	72.0	11.57
" 200	28.0	41.35	100.0	19.89

WASHED COAL FROM CLASSIFIERS

Mesh	Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
On 8	2.7	6.26	2.7	6.26
Thru 8 " 14	33.6	7.75	36.3	7.64
" 14 " 20	23.2	8.29	59.5	7.89
" 20 " 28	19.1	9.23	78.6	8.20
" 28 " 35	11.0	9.64	89.6	8.35
" 35 " 48	5.2	11.59	94.8	8.55
" 48 " 65	2.8	11.51	97.6	8.62
" 65 " 100	1.4	15.77	99.0	8.72
" 100 " 150	0.5	24.52	99.6	8.82
" 150 " 200	0.2	36.20	99.8	8.88
" 200	0.2	49.65	100.0	8.95

RHEOLAVEUR FEED

Mesh	Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
On 8	1.0	19.23	1.0	19.23
Thru 8 " 14	19.4	17.52	20.4	17.58
" 14 " 20	19.4	18.45	39.8	18.02
" 20 " 28	20.3	19.20	60.1	18.39
" 28 " 35	15.6	23.17	75.7	19.40
" 35 " 48	9.7	34.77	85.4	21.18
" 48 " 65	6.3	35.60	91.7	22.17
" 65 " 100	4.9	35.57	96.6	22.85
" 100 " 150	1.8	50.30	98.4	23.30
" 150 " 200	0.6	52.43	99.0	23.50
" 200	1.0	52.18	100.0	23.76

UNDERFLOW TO BE RE-WASHED

Mesh	Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
On 8	4.1	9.88	4.1	9.88
Thru 8 " 14	36.9	11.14	41.0	11.07
" 14 " 20	21.0	13.04	62.0	11.72
" 20 " 28	17.6	16.02	79.6	12.63
" 28 " 35	10.2	20.28	89.8	13.79
" 35 " 48	5.1	26.42	94.9	14.11
" 48 " 65	2.5	27.13	97.4	14.82
" 65 " 100	1.4	35.84	98.8	15.10
" 100 " 150	0.8	43.77	99.6	15.32
" 150 " 200	0.2	46.70	99.8	15.38
" 200	0.2	47.46	100.0	15.45

OVERFLOW RE-WASH SUMP TO WASTE

Mesh	Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
On 8	0.4	14.85	0.4	14.85
Thru 8 " 14	1.1	8.84	1.5	10.40
" 14 " 20	3.4	7.40	4.9	7.89
" 20 " 28	6.4	8.02	11.3	7.96
" 28 " 35	10.9	9.63	22.2	8.76
" 35 " 48	15.8	12.02	38.0	10.15
" 48 " 65	20.3	17.77	58.3	12.85
" 65 " 100	13.4	23.50	71.7	14.84
" 100 " 150	7.6	36.39	79.3	16.87
" 150 " 200	20.7	49.78	100.0	23.68
" 200				

REFUSE TO WASTE

Mesh	Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
On 8	2.9	43.10	2.9	43.10
Thru 8 " 14	19.5	45.26	22.4	45.0
" 14 " 20	13.9	47.02	36.3	45.8
" 20 " 28	14.4	49.98	50.7	46.8
" 28 " 35	13.8	54.62	64.5	48.7
" 35 " 48	11.2	61.44	75.7	50.7
" 48 " 65	9.3	60.93	85.0	51.7
" 65 " 100	7.9	53.94	92.9	51.9
" 100 " 150	3.5	62.08	96.4	52.1
" 150 " 200	1.3	59.90	97.7	52.3
" 200	2.3	58.92	100.0	52.46

COAL END TO DEWATERING SHAKER

Mesh	Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
On 8	1.5	7.01	1.5	7.01
Thru 8 " 14	32.4	7.85	33.9	7.82
" 14 " 20	21.2	9.34	55.1	8.39
" 20 " 28	18.6	10.42	73.7	8.92
" 28 " 35	12.5	13.55	86.2	9.60
" 35 " 48	6.3	16.27	92.5	10.03
" 48 " 65	3.8	19.57	96.3	10.42
" 65 " 100	2.5	26.55	98.8	10.81
" 100 " 150	0.8	35.89	99.6	11.00
" 150 " 200	0.2	44.36	99.8	11.05
" 200	0.2	46.39	100.0	11.16

SHAKER DISCHARGE COAL TO MARKET

Mesh	Percent Size	Percent Ash	Cumulative % Size	Cumulative % Ash
On 8	2.1	6.46	2.1	6.46
Thru 8 " 14	35.4	7.87	37.5	7.77
" 14 " 20	27.0	9.01	64.5	8.27
" 20 " 28	18.6	9.06	83.1	8.44
" 28 " 35	8.9	9.73	92.0	8.58
" 35 " 48	4.0	10.43	96.0	8.63
" 48 " 65	1.9	14.06	97.9	8.75
" 65 " 100	1.7	19.88	99.6	8.95
" 100 " 150	0.2	31.32	99.8	9.10
" 150 " 200	0.1	39.85	99.9	9.13
" 200	0.1	49.21	100.0	9.16

The Hudson Coal Company at Marvine Colliery have two methods for the preparation of their fines; one plant is made up of one Dorr thickener, 12 Deister-Overstrom tables and 4 classifiers. This plant prepares all the fines, minus 3/32-in. made at Marvine Colliery. The companion plant is a Rheolaveur plant with dewatering shaker on coal end.

The Dorr plant receives the wash water fines in a 25-ft. diameter hydroseparator, the overflow to waste, underflow distributed directly to 12 tables, amounting to 3½ tons per hour per table, the coal ends of each three tables to one classifier. The slate ends from the tables and classifier overflow to waste. Rake product prepared unsized fines, amounting to about 2.5 tons per hour per table.

The Rheolaveur plant at Marvine handles the fines from other collieries de-

(*Continued on page 52*)

Speeding Up Rock Work in Anthracite Mines

By Russell L. Suender*

As early as 1916 anthracite operators undoubtedly interpreted a handwriting on the wall that indicated storms ahead for the anthracite industry. They realized that substitute fuels, bituminous coal, coke, oil, and gas would compete with anthracite in increasing quantities for domestic heating and for industrial use, and that this competition would be all the more severe if anthracite selling prices could not be held within reasonable limits. At the same time they had to deal with a strong labor union, aided in all controversies with vote-seeking politicians. The climax was reached in the strike of 1925 and 1926, following the strikes of 1920, 1922, and 1923. Following the strike of 1925 and 1926 a new era in anthracite began, and for the first time in a decade the operators felt that they had an arrangement with the employes that enabled them to build with confidence, regardless of the then immediate outlook.

"Necessity is the mother of invention." Selling prices were too high and had to be reduced. Wages were fixed and most difficult to reduce. The engineering brains of the management in the industry was called upon to lower costs, and unquestionably much has been accomplished in this direction. Anthracite management, faced with this necessity of lowering costs, struck out in every direction, and stripping and rock contractors soon realized that their work had to be done at reduced costs. They accordingly busied themselves on the problem, with the result that machinery and equipment manufacturers soon learned that the anthracite territory, regardless of the depression in the industry, was one of the bright spots in the country to increase sales of equipment and machinery of all kinds. The days of slow speed in development work were past; what the operators demanded was the speeding up of rock work and the immediate mining of the coal developed in lieu of the slower methods of rock development, and delayed mining of coal. This is now being accomplished in the average anthracite mine at a material saving in expense. In other words, industry in general, including the anthracite industry, faced hand-to-mouth buying by the public and was forced to adopt a policy in keeping with it. So the slogan today is, "Drive your rock tunnels and develop new areas when you must, but when you start the job do it with dispatch."

Time was when rock tunnels were drilled with the old type hand jumper—progress was naturally slow (25 to 30 ft. per month) and costs and capital tied up were tremendous. In this business of mining increased costs for a fixed production is natural because the working faces are extended and on account of the smaller seams now mined. This must be offset by the ingenuity of management. Thus small and later large air compressors came into use, along with the replacement of the well-known mule with various methods of more effective transportation. Conditions were now such that development work in coal and rock could be speeded up to the great advantage of the producers, and in turn the consumers of the product.

To meet competition in bidding for rock work, tunnel contractors were forced to purchase the latest type of not only drilling but loading equipment, for the time had arrived that hand mucking of the tunnel faces had to be discarded on all long tunnels and on shorter tunnels where speed was a necessity.

In anthracite mines it had been the custom for many years to drive gangways even in the thick veins. The mining of the coal in such gangways required their maintenance for periods of 10 to 25 years. Under such conditions timber maintenance, especially after three to five years, became a very serious problem; in fact, so serious that today rarely does one find coal gangways in thick seams, and the tendency is to drive all coal gangways only where such gangways can be robbed back and abandoned within the life of the original gangway timber, or three to five years. Now this was made possible only because the cost of rock work was materially reduced and the speed of advancing faces materially increased. Today you find throughout the steep-pitch anthracite mining fields rock gangways driven in solid rock or leader veins underlying the thick veins for the full length of a property, and the coal developed and mined either through panel tunnels at approximately 600-ft. centers or by the rock-hole method (approximately 50-ft. centers).

Management was quick to realize that for a material increase in first costs, rock gangways underlying the thicker veins permitted quicker development and concentration of tonnage, all resulting in very much lower ultimate costs and greater recovery of coal, which latter item is of tremendous importance in the final results of operation.

Because of all this racket started by management in the anthracite mines, the rock contractors must not only get on their toes but they have to stay there or go out of business. The progressive contractor must be prepared to tackle from the smallest to the largest of jobs—tunnels 30 ft. to tunnels 15,000 ft.—rock gangways thousands of feet long, slopes in rock (not in coal as formerly driven) 250 to 1,000 ft. deep, and shafts 250 to 1,000 or more feet deep. For these jobs he must be prepared to furnish all of the drilling and mucking equipment, and in some cases the transportation, ventilation and compressed air installations. This latter requirement usually applies on tunnels, slopes or shafts that are not connected with active workings of the mine. Within reasonable limits we prefer to furnish all equipment and supplies, and operate the job as a distinct, separate unit from the mines—such an arrangement places responsibility for results where it belongs, and invariably means better speed. In my opinion, the speed of rock work today in anthracite mines is still retarded too much due largely to failure of service by the operating company in either transportation or continuous and adequate air supply of proper pressure.

Pneumatic rock drills, electric haulage, large air compressor installations are now in common use in anthracite mines, and mechanical mucking is making constant progress. This latter phase was the logical additional step necessary to speed up rock work; it eliminates corraling of hand-mucking crews, a very laborious work and a type of much undesirable labor, that is more or less intermittently employed; depending, of course, upon the amount of rock work under way.

Mechanical mucking pertains particularly to tunnel work where speed is of primary interest or rock gangways where at least two faces can be made for mucking each day of eight hours working time. That means the rock gangways must be at least on the same level and preferably turned off the same tunnel. For this application, and that of long basin tunnels or drainage tunnels which are of little use until completed, mechanical mucking is ideal. Due to the size of these tunnels, which are very seldom larger than 7 by 10 or 8 by 12 in. section, the type of equipment is necessarily limited. In the past few years the Hoar loader, now made by Allis Chalmers Manufacturing Co., the Butler Loader, Sullivan Scraper and Myers Whaley mucking machine size No. 4 have been used quite extensively. The Hoar and Butler both operate only on compressed air while the Sullivan and Myers Whaley although they can operate on air are by far more satisfactory with electricity as the source of power. With any one of these machines we have increased the speed of our tunnels from approximately 300 ft. per month to slightly over 400 ft. per month —an increase of at least 33 1-3 percent. This we have actually accomplished with both the Hoar and Myers Whaley machine and I understand from other con-

(*Continued on page* 52)

* Hill & Suender, Contracting Engineers.

Notes on MECHANICAL MINING in ANTHRACITE

By John C. Haddock*

Compiled from Records of
CADWALLADER EVANS, Jr.
E. S. CHRIST
E. L. DANA, Jr.
GEORGE B. JONES

WHEN the writer was first advised that he was to submit a paper entitled "Mechanical Mining In Anthracite," he refused, as he was extremely anxious not to fly under false colors with The American Mining Congress. In the first place, to cover intelligently the question of mechanical mining in anthracite would require so long a discussion as to become virtually a textbook, either with or without value and, in the second place, the accupation of president of an independent anthracite company today is only about 3 percent mining engineering—the remainder seems to be a combination of auditor, press agent, pawn broker, and taxicab. The good old days when a coal operator could sit in his office, negotiate with his men, look over next month's orders and sign dividend checks at quarterly intervals, have been past for some time. From the post-war period of investigation, we went from animation to beration, then stagnation, so that between constant efforts to cut costs in all departments of the business, repeated visits to one's customers to urge them to take coal, supervision of the innumerable reports that we make to the various tax collecting and other bureaus of our national and state government, one has scarcely more than an hour or two a day left to read over the weekly bulletins published by the statistical services proving how bad business is in the United States.

I trust I have, therefore, indicated how little time there is to become actually technically familiar, as a superintendent at the face daily, with mechanical mining in anthracite. What I propose to present to you is a digest of some of the research work and actual practical experimentation performed by such men of ripe experience as Cadwallader Evans, Jr., general manager for the Hudson Coal Company, one of our best informed and most progressive mining engineers and general managers. Joining with him in the submission of data are E. S. Christ, of Weston, Dodson and Company, Edmund Dana, Jr., an investigating engineer connected with our own main office, together with certain figures of George B. Jones, also with our company.

Before incorporating the statistics just mentioned in this paper, I want to place before you the actual physical condition of the veins which are to be mined mechanically in the anthracite coal field.

Of his personal knowledge, the writer knows of only one bed which, even for a comparatively generous area, maintains an ideal theoretical set-up for longwall mechanical mining. This bed is approximately 36 in. of solid coal, running, in a Channel sample, not more than 4½ percent of ash. It is overlaid by an exceedingly good roof, which does not show signs of oxygenation, or deterioration, for at least 48 hours after exposure to the air. It is underlaid by a bench of soft slate running from 3 to 5 in. in thickness. However, this particular piece of coal lies on a 30-degree pitch and there is some little doubt in the operator's mind as to the best way to handle propping and jacking to obviate the natural difficulties against breaking off the roof a suitable distance behind the face. Many other beds that might normally offer excellent opportunities of longwall mining in one of its forms, with at least part mechanical installation, are seriously handicapped by the fact that the thick coal adjacent to these areas, which were considered worthless up to 20 years ago, had been removed almost entirely or in part.

This has created a cave situation that makes it almost impossible to proceed in that orderly, scheduled manner so essential where mechanism is employed and so vitally necessary if any measure of financial profit is to be obtained therefrom; neither must we forget that a large number of anthracite coal measures are now beneath cities of the second and third class. In fact, certain areas of one of the largest cities in Pennsylvania have depressed from 12 to 14 ft., together with brick buildings, churches and other substantial edifices erected thereon, because of the removal of coal under them. It is quite the usual thing to find in our leases a restriction that sufficient pillars be left to support the surface and even in cases where this is not present, common sense and humanity prevent the operator from creating such surface damage that he endangers the lives of human beings from broken gas pipes and water mains on general and terrific disturbances.

These points are mentioned to you so that you will understand mechanical mining in anthracite is not all based upon driving a longwall gangway, installing shaker chutes, loaders and undercutters with the coal moving in an orderly manner to mine cars and thence to the tipple. To sum up, let us imagine mechanical perfection in an anthracite coal mine. This would comprise mechanical undercutting of the coal, highly mechanized drilling and shooting, mechanical transportation to the car and thence to the surface. We find that from the car on, our present technique is so well mechanized as to substantially reduce labor cost, but from the car back, we have the difficulties which I mentioned to you above, coupled with the obvious peculiarities of the anthracite bed structures themselves.

I am, therefore, presenting to you the following:

1. What is termed a successful technique of mechanical mining as used by Cadwallader Evans, Jr., in the fairly level bed structures operated by the Hudson Coal Company in the northern anthracite field, and,
2. What seems to me to have been a sound technique overcome by adverse general conditions in the Beaver Brook Colliery, under the jurisdiction of E. S. Christ, that had to be discontinued, and,
3. A theoretical basis of procedure that our George B. Jones hopes to employ in a certain bed at one of our collieries under development in the southern field near Pottsville, Pa., as tabulated and presented by Mr. Dana.

I will make no comment upon them for the very simple reason that I feel the ideas which have been developed and the study which has been given to the matter by the gentlemen in question, have so far exceeded any possible experience of my own that criticism would be unjustified and we will, therefore, proceed with them.

By CADWALLADER EVANS, Jr.

SHAKER CONVEYORS IN CHAMBERS

AT THE Stillwater Mine of The Hudson Coal Company, 25 miles north of Scranton, the Clifford Bed, averaging 32 in. in thickness with a 5-in. band of unmarketable bone in the bottom, is being undercut and loaded by means of shakers. The immediate roof is from 2 to 6 ft. of hard, black, slatey rock, and the remainder of the 125 ft. of overburden is fairly hard sandstone. The roof is strong, except for occasional local rolls, and the bed lies on a pitch of from 2 to 8 percent.

The gangways are driven along the strike, with chambers to the rise. Shaker conveyors were first installed in June, 1929, and have resulted in increasing the output in tons per man as compared with the scrapers previously used. The crew consists of four men; one stationed on the gangway, handling and topping cars, and the remaining three at the face. These men undercut the coal with the machine, drill, fire and load. The man stationed at the cars is utilized in propping and handling supplies, when no loading is going on. This crew undercuts and loads from two to two and one-half 6-ft. cuts per shift; each cut yields 16 tons. (*See tabulation—Part 1.*)

* President, Haddock Mining Company.

Location	Equipment	No. of operations considered	Bed	Bed sections	Kind of mining	No. of operation starts	No. of men per operation	Daily production (tons)	Total production (tons)	No. of tons per man shift
			Part 1							
Stillwater	Shaker conveyor	3	Clifford	Coal—27" Bone—5"	Cha.	982	4.3	35.5	34,941	8.2
Stillwater	Scraper loader	2	Clifford	Do.	Cha.	226	9.3	61.5	13,899	6.6
			Part 2							
Jermyn	Shaker conveyor	2	Dun. No. 3	Coal—29" Rock—9"	Cha.	312	4	29.5	9,201	7.3
Jermyn	Scraper loader	2	Dun. No. 3	Do.	Cha.	72	10	61.0	4,423	6.1
			Part 3							
Stillwater	Shaker conveyor	1	Clifford	Coal—27" Rock—5"	Longwall	14	21.3	196.0	2,744	9.2
Stillwater	Scraper loader	1	Clifford	Do.	Longwall	56	13.6	116.0	6,515	8.6
			Part 4							
Olyphant	Shaker conveyor	1	Rock	Coal—6' Rock—3'	Pillar Robbing	56	5	36.8	2,060	7.4

A similar system of working is in use at the Jermyn mine, where the Dunmore No. 3 Bed is being worked with undercutters and shakers. This bed averages 38 in. and is in two benches of coal with an 8-in. band of rock separating them. The roof is of hard sandstone and the cycle of operations is similar to that at Stillwater described above, but the output is slightly lower, due to the handling of extra rock in the bed. (*See tabulation—Part 2.*)

Shaker Conveyors on Longwall

The first installation of shaker conveyors on longwall was made in the latter part of 1930 at Stillwater operation in the Clifford Bed, which is 32 in. in thickness, with a 5-in. band of unmarketable bone at the bottom. In preparing a longwall face, the chamber is driven from the gangway up the pitch a distance of about 270 ft., and is timbered systematically with props stood on 3½-ft. centers, but with the first row of these props held 6 ft. from the face which is to be developed as a longwall face.

The shaking conveyor is installed between the inside row of props, and this conveyor is about 1 ft. away from the props, allowing 36 in. between the conveyor troughs and the solid working face which is sufficient clearance for the longwall undercutter. The average depth of the undercut is 5½ ft.

This operation works two shifts, the loading all being done on the day shift and the cutting, moving, propping and firing on the night shift. The total force varies from 21 to 23 men; 12 men on the loading shift and the remainder on the deadwork shift. The 12 men on the loading shift require from 6 to 7½ hours to load out the whole of the coal undercut at the face.

Due to extremely hard undercutting, it is necessary to work the undercutter for a shift and a half, and the undercutter, therefore, starts to work at 10 o'clock in the morning, and works through until the job has been finished at about 11 o'clock that night. (*See tabulation—Part 3.*)

The figures given show that with the shaker conveyor a larger output per man has been obtained than was possible with the scraper loader.

Shaker Conveyors in Pillar Removal

An example of this is shown in the Rock Bed at Olyphant Colliery, which bed is 9 ft. thick, containing about 3 ft. of refuse in four separate benches in the center of the bed. The roof is of sandstone and the pitch of the bed varies from 8 to 25 percent in favor of the load.

First mining in the territory where the shakers are working was done many years ago with the roadways laid in the center of the chamber and the refuse gobbed along both ribs. Since first mining was completed, the roof rock has broken down so that the chambers also are now more than three-quarters filled with gob and broken roof rock. The pillars are narrow—15 to 18 ft.—but it is possible to take 6 ft. of skip advancing up the pillar, using the shaker conveyor with very little handling of gob and roof rock.

On each pillar the force varies from 3 to 6 men per shift and the work is done double shift and occasionally triple shift, in order to get out of a given pillar before squeeze occurs. The tabulation (*Part 4*) shows the results obtained.

By E. S. CHRIST

OUR activities at Beaver Brook were in the Gamma or Seven Foot Vein. Drilling records had shown this to be a 42 to 48-in. all coal seam, whereas in actual development it proved to be approximately 50 percent bad bone, existing in benches varying from 2 to 6 in. in thickness. One 6-in. bench started about 1 in. above the bottom rock, and a 4 to 5-in. bench carried next the roof, the others being interspersed between. It was this bony which prevented successful performance. It was necessary to do our undercutting in the bottom bench, which was as hard as sandstone, and which frequently made cutting almost impossible. From my notes, I give the following data taken by myself, to illustrate our cutting difficulties, recorded January 10, 1928:

	1st Record 1 p. m.-5 p. m.	2nd Record 5 p. m.-10 p. m.
Elapsed hours	4	5
Elapsed minutes	240	300
Minutes cutting	69	77
Minutes changing pick points	171	223
Pick points changed	111	114
Feet of face cut	25'-5"	7'-0"

You will note that in nine hours we were able to undercut but 32 ft. 5 in. of longwall face, using 225 pick points, plus the original 32 in the machine, and requiring 394 minutes or 73 percent of the time. Ordinarily a 200-ft. face should be cut with one change of picks (32), which, plus the original set, would total 64 picks or about 3 ft. per pick. From the above time you will note we cut an average of about 1½ in. per pick used.

It was impossible to jump this lower bony bench and cut in a coal seam, as this bench did not form a good footing for the roof jacks, due to being underlain with a small seam of coal next the main bottom.

The seam of bony next the roof also presented a problem in loading. This bench shot off in large pieces, some having an area of 4 ft. by 6 ft. This naturally reposed on top of the coal pile after shooting and had to be broken with sledges before loading could be started. The low vein, and proximity of this bony to the roof, made breaking and loading very arduous. It could not be gobbed, for, in longwall work where roof jacks are used, the jacks must be constantly advanced from a rear position toward the face and consequently must have an unobstructed path.

The foregoing is advanced to support my contention that both performance and costs were far from what might be expected under normal conditions, both from the viewpoint of dollars expended and the equally damaging low coal recovery per mine car loaded. In addition, we encountered some spots where vein was as low as 21 in. with side pitches and adverse pitches. There were some small minor basins which accumulated water, making for difficulties in loading and pack moving. Our best load-

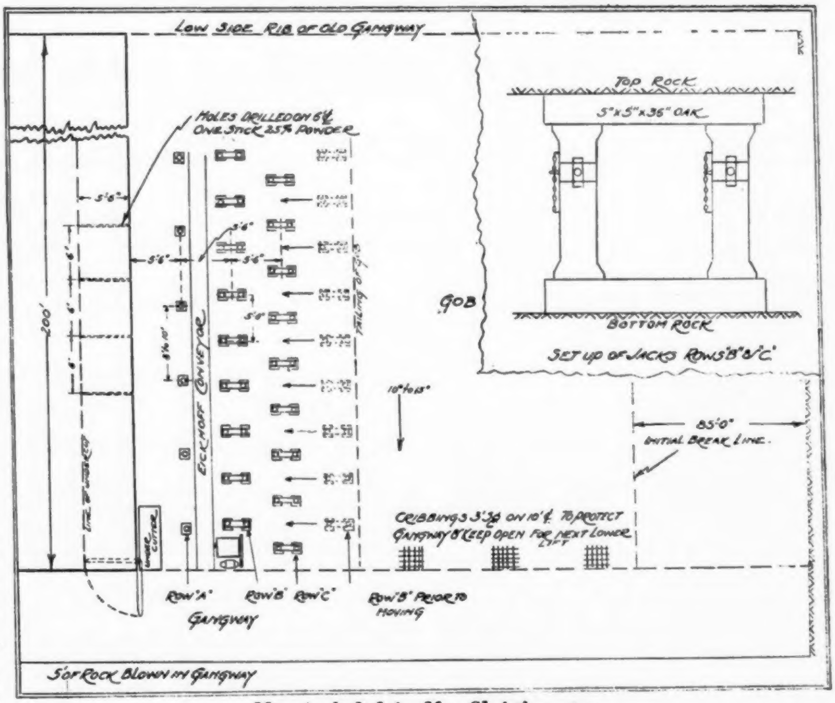
ing record (which should have been our usual one) was 64 cars in eight hours.

The actual cycle of work is theoretically as follows:

3 P. M.-11 P. M. Face has been cleaned out and prepared for cutting. The cutting machine has been brought down from the upper end of the face and the runner has sumped his cutter bar in just off the gangway. The cutting machine requires an operator and helper. The latter shovels the tailings into the conveyor and assists in changing pick points, etc. The Jeffrey machine which we used cut to a depth of 5½ ft. and the cut was about 6 in. vertical. The tailings ran almost 50 percent prepared sizes. The machine dragged itself along the bottom rock by means of a rope winding around a ratchet feed drum, the other end being fastened to a jack up the pitch. As mentioned, these men should cut 200 to 250 ft. of face in eight hours. Upon completion of cutting, the machine was moved over from the top of the wall sufficiently to prevent damage from shooting.

3 P. M.-11 P. M. While the face was being cut, a crew consisting of a jack boss and three jackmen moved the rear row of double jacks and cap pieces a distance of 11 ft. From the sketch you will notice the position of these jacks. Row "A" is a single row with oak cap and foot pieces located 5½ ft. from the face before cutting commences. Row "B" is a double row of jacks advanced from the dotted position and comprising the set, the moving of which we are describing. Row "C" is also a double row staggered with Row "B" and will comprise the set to be moved by the jackmen on the succeeding night. These jacks are placed on about 5-ft. centers up the pitch. The cap and foot pieces are 5 in. by 5 in. by 36 in. oak, made by cutting mine ties in half. The jacks are known as Langman Jacks made by the Lorain Steel Company. The ones we used were about 25 in. high in normal position. By combining two top pieces and a wedge we could make a 15-in. jack, and using two bases and a wedge we could get a 35-in. jack. Total weight was approximately 175 lbs. In removing these jacks, the key of the wedge is tapped with a hammer from a vertical to horizontal position, at which time a slight tap on the wedge will cause it to slide out of position and collapse the unit. Where pressure is great, the wedge actually shoots out of place, for which purpose a retaining chain is hooked through between top and base pieces. The jack is set up in its new position, and tightened with the use of a long handle wrench applied to the wedge bolt. The expanding plane surfaces of the wedge draw it up tightly in position. Where roof height varies, filler pieces are placed between the upper and lower faces of the jacks and the cap and foot pieces. The cost of our jack was about $11 each. It is very important that these jacks be set at right angles to the pitch, which in our case was from 9° to 15°.

11 P. M.-7 A. M. A miner and helper drill 5½-ft. holes on 6-ft. centers as closely to the roof as possible. One stick of 1⅛ in. by 8 in. 25 percent coal powder is placed at the back of each hole and well tamped. It is advisable to shoot the face in two sections to avoid danger of overburdening the battery. This work is completed by 7 a. m. in time for the day shift to load.



Map included in Mr. Christ's notes

7 A. M.-3 P. M. During this period, the loaders shovel the coal into a conveyor of the Eichoff or Cosco shaking type. The number of loaders depends upon length of face, thickness of vein, etc. Under normal conditions 8 to 10 cars per man would not be expecting too much. Upon completion of shoveling, these loaders advance the single row of jacks 5½ ft. toward the face or 5½ ft. from the face. They also advance the conveyor 5½ ft. using post pullers at four or five points and sliding the conveyor in its entirety to position for the next day's loading. Upon completion of this phase, the cycle is complete and the machine is again ready to be moved down the wall and resume the next cut.

Summarizing, the following force would be required, presuming a 48-in. seam of good coal with fair regularity:

Occupation	Day rate
1 General foreman	$9.00
1 Conveyor operator and car spotter	5.45
8 Loaders	43.60
1 Cutting machine operator	6.00
1 Cutting machine helper	5.45
1 Jack boss	7.00
3 Jack men	16.35
1 Miner (drilling and firing)	6.00
1 Laborer	5.45
18 Total	$104.30

This comprises only face employes, not including transportation, electrical maintenance, sharpening of cutter picks, etc. Nor does it include cost of power, explosives, compressed air, etc. A 200-ft. face with 4-ft. coal vein should produce at least 60 cars of 100 ft. capacity, or 150 tons per cut, so that under normal conditions the face cost would be extremely low.

The initial investment is considerable. It includes driving of gangway with attendant blowing of rock, driving of initial chute; driving of a safety heading at the top of the face and paralleling the gangway (unless an old upper gangway is available); purchase of conveyor, cutting machine, jacks, fan and ventube, etc.

The initial break usually occurs following removal of about 80 to 85 ft., or approximately the fifteenth cycle. Breaks thereafter will depend upon character of roof. We found the breaking of the roof followed the moving of jacks with fair regularity after the initial break had been made.

The system makes for concentrated supervision and should result in few compensable cases. In about one year of operation we had but one case, which was where a man attempted to turn a wedge key with his hand and nipped off the top of his fingers.

THE PROPOSED PLAN OF DEVELOPMENT AT THE SALEM HILL COLLIERY

By E. L. DANA, Jr.

WITHIN the last decade four or five revolutionary machines have been developed for the mining of coal. Wherever these machines were installed, they necessitated such marked changes in the former methods that they merited classification as a new mining system. The name applied to this new class may unfortunately lead to some confusion, unless the reader is introduced to the precise meaning of the word in this application. "Mechanical Methods," in a lit-

eral sense, has a very broad meaning, including every piece of machinery employed, from the electric time clock to the locomotive. As used with reference to the mining industry, it has come to have a narrowly restricted connotation: *It refers to the replacement of hand labor by machines* (pit car loader and drilling machine) *and includes the new type of manual labor arising from the use of the machines.* "Mechanical Mining," as used in this paper should be assumed to be employed in the above defined local sense.

Before going into a discussion of the new methods at Salem Hill Colliery, the subject of mechanical mining will be divided into four major functional classes —(1) drilling, (2) undercutting, (3) loading or mucking, (4) transportation. These classes need no definition, their limits being very nearly contained in their titles. Our examination of the Salem Hill property will be made with reference to each one of these classes in the above order.

The Salem Hill property includes a section of a basin running east and west under the Schuylkill River, the rise of the basin running in a hill immediately bordering the river. The tract extends for about 5,400 ft. in the direction of the axis of the basin and, perpendicular to the axis, to the outcrop of the upper coal measures. In all it represents a tract of roughly 200 acres.

The coal lies in four beds, Salem, Rabbit Hole, Tunnel, and Black Mine, varying from 2½ to 8 ft. in thickness. The bottom condition may be described as good, but the top is at many points a weak, inelastic sandstone.

Development to date includes a horizontal haulage tunnel driven north into the hill perpendicular to the axis of the basin and line of strike, a slope down from this tunnel on the south dip of the Rabbit Hole vein, gangways driven east and west on the line of strike at five lifts, and tunnels driven horizontally north and south into the underlying and overlying beds at these lift levels.

Longwall mining will be attempted wherever possible, though it is not unlikely that the uncertain condition of the top will give rise to roof control problems that will interfere with this program in the thicker veins. We shall start out mining the thin veins with the longwall method, will apply it to the thicker veins, if possible, and, if not, will complete the work by means of breast mining, "V" mining, or some like system of hand work. Bearing in mind the natural conditions and state of present development, an examination of each division of mechanical mining with reference to the Salem Hill Colliery is now in order.

The first section, *drilling*, has undergone radical changes within the last few years. New considerations, placing the muck in the best position for mucking by machine, attempting to break the cut into the size of lump best suited to mechanical mucking, speed in the drilling operation, etc., have made necessary new drilling methods in all parts of the mine. There are many new devices for power drilling, ranging in complexity and price from the small hand portable type to the heavy machines which are motorized to run on the gangway track.

Wherever there is a longwall operation at Salem Hill, it is planned to use a specially trained drill crew, either with a drill mounted on the cutting machine or a portable power driven installation. We do not expect to use a drilling crew in panels mined by breast mining for, in relatively thin beds, more than one cut can be taken per shift, making it impossible to do all the drilling in the night shift or shift the crew fast enough on the day shift. For this reason, the training of a drill crew for breast mining will not be attempted.

The second division, *undercutting*, is coming to be regarded as more and more desirable from every viewpoint, the possibility of better control of the cut, less shattering of the coal, very little and sometimes no explosive being necessary in this method, more complete and pure recovery of the coal; all these things make undercutting advisable wherever possible.

This system will be put in practice in all the longwall faces, using a low, self-propelled machine, which pulls itself up the pitch. A certain amount of difficulty is anticipated if we encounter vein thickness temporarily less than the machine height, but these conditions appear in only a few points in the measures as exposed in the existing gangways.

The third division, *loading*, is, in the case of Salem Hill, greatly simplified by the pitch on which the beds lie. Coal can be removed from the face, transported to the gangway, and loaded into cars by means of gravity alone. Although this eliminates the necessity for face loaders, shaking chutes, scrapers, ets., in actual mining of the coal, the problem is still present in development; I. E., gangways driven on the strike. Here the cleaning up will be done by some type of mechanical car loader, either of the belt type or the scraper and slide.

The fourth division, *transportation*, under which is here classed all carrying, with the exception of the immediate removal from the face, is done by mine cars, gangways, slopes, and tunnels. This particular method of transportation presents some obstacles to development work, particularly in a gangway in which mining is being done. In the removal of the loaded cars and the spotting of empties, considerable time is wasted in shifting the cars, unless more than one car can be loaded without shifting the trip.

Mr. Jones, the superintendent at Salem Hill Colliery, has devised a plan which he hopes will enable us to load three or four cars without moving the trip: Using a pit car loader of the belt type, the horizontal section is extended over the cars the distance required to cover three, four, or as many cars as you wish. This extension, in the operating position, rests on stilts to the gangway floor or bottom. When it is desired to move the loading position, the legs are removed and the extension rests on the cars, being moved in that position. Provision for demountable sections should be made in building this extension for gangways in which severe curves are encountered. With this device, three or four cars could be loaded before the gathering locomotive would be required to replace the cars with empties.

The foregoing is almost entirely conjecture, contingent on future verification of the conditions which appear to exist in what beds are now exposed to view. Practical difficulties may very possibly force us to modify, or even discard, the plan as it has been set down.

There is, however, one phase of mechanical mining which has been tested at the Salem Hill property. The earlier gangways and tunnels were driven as contract work with the well known hand methods. Within the last year mechanical methods were applied to this work, with very satisfactory results. In this gangway, company labor was employed, the drilling was organized and trained, drilling was done with jack hammers, the cleaning up done by a scraper and slide loader, and close supervision was given to all the work. With this system, including the interest on investment, depreciation, and repairs to equipment, it was possible to show a consistent saving of 50 percent over the cost per yard of driving by the old method.

The above saving resulted from factors which we feel will make mechanical methods successful in our other work: *Increased output per man, less wasted time* of the other men and the equipment, *better organization* and *supervision*. This last is perhaps the most important and it might appear that hand mining, given the same close supervision, would then compare more favorably with the mechanical mining. This is undoubtedly true, but the scattered nature of hand mining, as contrasted with the great possibilities for centralization inherent in the mechanical mining, conclusively shows the difficulty of adapting the old mining methods to new standards of supervision.

The most significant change wrought by the introduction of machines, can be appreciated by inquiry into the duties of labor now employed in mining; whereas before the miner used to perform the complete cycle (drilling-firing-loading), there now exists a nice division of labor. Each man has well defined duties in a highly integrated process, a condition ideal from the standpoint of supervision and training.

In conclusion I wish to bring to your attention a matter which is very close to me and has little to do with mechanical mining, except indirectly. Coal has been one of the fundamental industries in our nation for generations. In my personal opinion, it will continue to be, but it will only continue to be by going hand in hand with the mechanical engineer and the chemist. We must remember that mechanism will decrease costs and costs must be continually reduced. Mechanism in turn saves the public trouble, and the public today buys that which creates the maximum of comfort and the minimum of trouble.

I really have not placed before you today a definite theory involving mechanical mining in anthracite. I have given you a series of circumstances which govern mechanical mining, together with three or four tentative solutions—one of which, at least, is successful. Take our problem away with you and if, with the vast experience present, any one of you have suggestions to offer or theories to develop, by all means keep on with them, because the anthracite industry needs the advice of others and I can assure you will appreciate it.

JOHN C. HADDOCK.

Cost of MINE ACCIDENTS

By R. M. Lambie*

HIGHER conception of our obligations to serve humanity and wider recognition of the certainty of continuous operation of the laws of economics have been the largest major factors in the development of the recent national trend toward safety in industry.

To these two fundamental reasons may be attributed the growth and expansion of the safety movement in the diversified industries which have made America the foremost industrial nation of the world.

It is not my intent to discuss the cause of safety from the humanitarian viewpoint, yet I take this opportunity to say that in the coal-mining communities where we have sought to enlist the mothers, wives, and children of the employes in the safety cause, our direct appeal to dependents of workmen has contributed in no small measure to decrease the toll of accidents. When men are made to realize that others suffer from their carelessness or recklessness, there is usually a cessation of their defiance of known hazards.

Through the organization of safety clubs in mining towns, holding regular meetings at stated times and attended by the families of employes and employers, there has been developed a town or community spirit where safety in employment has been emphasized, although other subjects of community interest may frequently be the topic of discussion. These frequent meetings of safety clubs strengthen the morale of the community and give men and management an opportunity to share in the solution of community problems.

While, in my opinion, the humanitarian aspects of the safety cause are fully appreciated by both employer and employe, I am convinced that the economic phase of the safety problem, for many years disregarded by executives in industry, is now recognized by efficient industrial management as a major objective which can not be ignored.

This belated recognition of an economic obligation has been forced by a consideration of cost sheets. The wide sweep of workman's compensation legislation since 1910, now embracing 44 of the 48 states, brought to industrial management a visualization of the cost expenditures that was formerly obscured in a maze of litigation that challenged accurate or even approximate computation.

Work injuries formerly were thought to be uncontrollable in industry and regarded as a matter of expectation in the operation of mines. Systematic compensation for work injuries, which had its genesis in Germany in 1884 and is now almost universal in scope, demolished the belief that responsibility for work injuries could not be allocated. These statutes afford relief for accidents in terms of wage loss and give substantial assurance of protection to the workman and his dependants. The employe surrenders his right to sue for the recovery of damages for the established wrong, and the employer is divested of his ancient rights of defense when he becomes a limited insurer for work injuries occurring in the industry which he controls and operates.

Coincident with the enactment of compensation laws, employers began to strive for accident prevention. The idea took root that accidents were preventable and that every injury was the result of some maladjustment. Our industrial executives found that the direct cost of accidents could be easily computed, and they have found that industrial efficiency can not be attained without safety in operation.

There is an accumulation of indisputable evidence that industrial executives, where safety has been made a major objective, have lowered the frequency of accidents, and reduced the economic toll that follows in their wake. It is my pleasure to quote from the records of a great trunk-line railroad.

The Norfolk & Western Railway traverses my own state of West Virginia. Coal constitutes 85 percent of its freight tonnage. From 1912 to 1929 the number of fatal accidents on this road fell from 61 to 13; the number of injured persons from 2,675 to 601. The ton-miles per casualty on that railroad increased from 3,372,413 in 1912 to 28,440,665 in 1929, and the casualties per billion ton-miles fell from 296.52 in 1912 to 35.16 in 1929. It is an amazing record of progress and efficiency. It has been duplicated and even surpassed by some of the larger iron and steel organizations, some of which have enjoyed lower compensation rates than concerns which are devoted wholly to agricultural pursuits.

In the coal-mining industry there has been a commendable improvement in the reduction of preventable accidents. It has not been so apparent as the record made in some other lines of industrial endeavor. For a number of years, however, the coal industry has been engulfed in a whirlpool of depression, and this condition has had a baneful effect on both men and management in striving to promote the cause of safety. The realization from sales of coal at the mines has been so meager in recent years that improvements and the elimination of natural hazards have, in some cases, been indefinitely delayed. Yet, with all this, there has been a distinct improvement in the elimination of preventable accidents as operators have realized the staggering losses to which the industry has been subjected in direct losses without a consideration of the higher indirect losses they have sustained.

Last September, before the International Association of Industrial Accident Boards and Commissions, H. W. Heinrich, of the Travelers Insurance Company, estimated, on reports made by the engineering and inspection division of that company, that the concealed or indirect cost of accidents was four times as great as the compensation awards, including medical benefits. This estimate was based on research in 10,000 cases, and its accuracy has been frequently demonstrated by application of specific plants.

This indirect cost has been computed to include the following:

Wages lost by injured employes, time lost by other employes, time lost by superintendents and foremen, property damage sustained, cost due to interference with production, cost of subsequent injuries due to weakened morale.

Just as impressive as the explanation made by Mr. Heinrich were the forceful illustrations made by Mr. Rush N. Hosler, superintendent of the Pennsylvania Compensation Rating Bureau, last December before the Coal Mining Institute of America. In a study of the cost to the bituminous coal industry of that state, Mr. Hosler showed that the direct cost of accidents in compensation and medical benefits during the five years 1924-1929 exceeded $25,000,000, or 3.6 cents per ton on every ton of coal mined in that period. In addition to this direct cost, he estimated an indirect cost of $5,000,000 per year, based on the stoppage of work, weakened morale, property damage, etc., a wage loss of $10,000,000 per year on account of injuries and a loss of time and consequently production of 17 man-years per year.

In the state of West Virginia, during the period 1921-1930, inclusive, the direct cost to the coal industry of workmen's compensation insurance, including medical benefits, has been $28,874,837, while all other subscribers to the compensation fund paid premiums of $11,569,299. The direct cost of compensation to the coal industry was based on wages paid of $1,465,000,000. The wages paid by other subscribers in this 10-year period amounted to $1,404,000,000. The coal production of the state during these 10 years was 1,204,170,059 tons. Eliminating the tonnage of self-insurers from this production, the average direct cost of compensation of the coal subscribers to the compensation fund of this state over this 10-year period approximates 2½ cents per ton. It has increased in recent years to approximately 3 cents per ton.

Using the Heinrich and Hosler formula for computing the indirect or hidden cost of work injuries, it is safe to assume that the actual cost of mine accidents to the employes and employers of West Virginia is in excess of $100,000,000 for that

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* Chief, West Virginia Department of Mines.

SAFETY and MECHANICAL MINING

By W. J. Jenkins*

IN DEALING with the subject assigned to me, it is not my intention to discuss it from a strictly technical or engineering standpoint. Rather, I am going to present it from the viewpoint of an executive, who, in the last analysis, must stand the brunt as to cost, as well as answer the criticism of the injured employe and those dependent upon him.

The introduction and use of mechanical equipment, whether it consists of trolley and storage battery locomotives, chain or punching coal cutters, coal loading devices, power drills, etc., in and of itself, does not entirely remove the hazards of the occupation or insure absolute safety to our employes. It may be that additional safeguards are required. If and when guards and shields are provided, it then becomes necessary to intelligently direct all those utilizing mechanical appliances to avail themselves of these protective features.

It is the duty of the management to insist upon the use of the safety features supplied. Employes disregarding the laid down rules for safety should be cautioned, and if necessary, disciplined.

The introduction of any mechanical appliance with which the employes are unfamiliar usually carries along with it the idea that we must anticipate an increase in the number of accidents, by reason of the user of the equipment being unfamiliar with its operation. To this thought I do not subscribe. I do agree, however, that the efficiency or results secured in the operation of the appliance during its introduction, in many instances, is very disappointing. During such period the management should be patient, and not until the operator feels at home or has familiarized himself thoroughly with the equipment should he stress the point of increased efficiency and better results. Pressing the employe for maximum results too early in his experience will cause him to become careless as to his own safety or the safety of those working with him.

It may be said that this is "old stuff." Granted, yet it is the essence of satisfaction in operating coal mines.

At our Mt. Olive mine, practically every item of its operation utilizes mechanical aids, undercutting, drilling, loading, transportation, etc. Power is supplied from our own power house. Motor generator stations are located in close proximity to the face. The A. C., delivered to the stations at 2,300 volts, is transformed into D. C. and reaches the working places with the maximum voltage permitted under our state laws. The usual precautions prevail as to having locks on the motor generator stations, explosive boxes, etc.

* President, The Consolidated Coal Company of St. Louis.

In the loading of coal mechanically at this mine all employes working with a loading unit are confined to a range averaging 14 rooms and 2 entries and 7 to 8 crosscuts. In other words, the loading machine and its crew, on operating days, move about in only 6 or 7 working places. The cutting machine crew of 2 men is, of course, working just ahead of the loading unit.

The supervising unit consists of one foreman to each two loading machines and cutting crews. The operation of the two loading units is in closely connected territory, so that it is not uncommon for him to visit every working place several times a day.

Before the machine cutting crews go into a room an inspection has been made to determine its safety, first, as to roof conditions, and, second, as to safety at the coal face, and particularly as to the possibility of a fall due to a possible coal overhang attributable to sticky top, etc.

Supervision also carries with it more frequent visits from the mine superintendent, the mine manager and such assistants as may be deemed advisable.

Compare these practices with those of previous years, under the hand loading or individual contract system. This mine then had more than 700 working places and six supervising officers, as against 197 working places today and ten supervisors. With hand loading the miner many times took an unnecessary risk. He would continue to finish loading his car, rather than stop temporarily, set a prop or take down an apparently dangerous piece of rock. This temporizing in making the working place safe does not exist while he is employed on the mechanical loading units.

With each loading machine crew there is employed two certified miners as face men. The duty of these men, primarily, is to provide a safe working place by the setting of timbers, taking down loose or hanging coal, slate or rock. During the period of loading they continually sound the roof, break down coal that might possibly "hang," whether due to poorly placed shots or for any other reason.

In loading coal mechanically there is the additional supervision exercised by the operator, of the loader and his helper.

The adoption by the Consolidated Coal Company of the mechanical principle of loading dates as of May, 1928. Several types of mechanical loaders, as well as conveyor type have been installed. An analysis of accidents by years, starting with the year 1927, at which time we were hand loading exclusively, shows conclusively that the introduction of additional mechanical appliances has not increased the number of accidents; on the other hand, the number has actually decreased, with a corresponding increase in tonnage produced.

At this mine every accident, no matter how trivial, is and has been made a matter of record. As an illustration, the mine produced 112,704 tons during the month of March, 1931, with 19 accidents reported. Of these 19 accidents, no time was lost on 9, 3 lost one day each. The remaining 7 were of a more serious nature, however. Six of the 7 had returned to work before the middle of April, and only one of those injured during March had not recovered and returned to take up his usual duties.

It might be of interest at this time to note that only 4 employes injured prior to April have not recovered sufficiently to return to work. Only one of these injuries can be definitely attributed to mechanical loading. The injuries sustained by these 4 men cover: No. 1, sprained ankle, due to falling over coal; No. 2, a double rupture, suffered by a main line trackman; No. 3, fractured ankle, occurred while wedging down coal, a piece of which struck the injured employe's ankle; No. 4, while pushing cars the employe was caught between cars at a time when empties were being placed on switch by storage battery locomotive.

During the year 1930 there was reported a total of 379 accidents; 333 of these occurred underground and 46 on the surface. Of the underground accidents 218 were non-compensable. Of the 46 surface accidents 38 were of minor importance and did not call for compensation.

Of the 42 classified segregations or lists governing this operation: 15 classes suffered no compensable accidents; 14 classes suffered one employe accident each; one class suffered two employes accident; two classes suffered four employes accident.

The following table of comparisons for the last four calendar years is enlightening on the subject under discussion, since during the entire year of 1927 hand load-

MINE NO. 15, THE CONSOLIDATED COAL COMPANY

Year	Tonnage Produced	No. of Accidents	No. of Compensable Accidents	Tons per Accident	Tons per Compensable Accident
1927	598,704	467	146	1,282	4,101
1928	693,604	495	177	1,401	3,919
1929	708,899	423	146	1,676	4.855
1930	975,391	379	123	2,574	7,930

ing prevailed. In 1928, beginning with May, the greater part of the tonnage was secured by hand shoveling onto conveyors. In 1929 the greater portion of the tonnage was loaded by mechanical loaders. The year 1930 represents a year's operations, using only mechanical loaders.

A study of these figures is convincing. As employers using mechanical loaders, we furnished safer working places to all employes, as compared to the conditions during the year 1927, at which time hand loading methods prevailed.

We are firmly of the opinion that the improvement would not have been possible were it not for the introduction of the mechanical loader, with the more intensive supervision possible under that plan of mining.

In closing, I want to simply state that we believe the men are happier since the danger ratio has been bettered, while, on the other hand, as an executive, I can join with the men, in that "our treasury" also reflects a decided improvement.

Mechanical Loading at Little Betty

(*Continued from page* 35)

curves of short radius. The gathering head is equipped with flights that take a side swipe on the shot-down coal as it is pulled to either side, or while being sumped into the coal. These flights dig into the coal pile, undermine it, and carry it back to the discharge conveyor.

The chain on the gathering conveyor runs in a similar manner as a chain on a shortwall machine and it is reversible so that it will load equally well while traveling in either direction.

The discharge conveyor is an ordinary double strand conveyor such as is used in practically all loading machines.

We ordinarily feed the gathering head straight into the face, then feed it sideways, cleaning up one side of the room, and then reversing it, cleaning up the other side. This allows us to keep our timbering close up to the face. The gathering conveyor can be swung at right angles to the track, and therefore the machine does not require practically any more space for operation than the space that is occupied by the coal when shot down. The conveyor hugs the floor and scrapes it clean, leaving very little bug dust or coal behind.

The coal is cut for these loaders with a Jeffrey 29-C Arcwall track cutter equipped with an 8½-ft. bar. The sweep of the gathering conveyor takes an arc in exactly the fashion of an arcwall cutter and therefore is ideal for loading in places prepared by this type of cutter. One cut from each of three to four places usually gives the machine a day's tonnage; the average yield per place being about 40 to 50 tons.

Three shots are used per place and the coal comes down in very good shape for a loader that can handle lump coal. The coal is lightly shot as the major demands of the market absorbing the product are for lump coal.

Three men are used in the operation of each machine. This crew consists of one operator, one helper and one man to look after the car under the conveyor and to change cars. Three men therefore make up the complete crew on the loader itself.

One machine has loaded as many as 139 cars or 209 tons in one shift of 8 hours, and we average consistently, month after month, 104 cars or 156 tons per shift per machine. The loader handles the coal with practically no breakage. The coal at Little Betty is not being shot as hard for these machines as for hand or pit car loading.

We were very fortunate in securing men who were interested in the operation of the machine and who used their heads. It was only two or three days after the first machine was installed that we came up to good capacity, and this was due to the cooperation of the men, and having a definite system planned for the men to use. Each man had a certain task that he learned, and no time was lost. At the same time the men did not have any such heavy labor as before.

For instance, and to show how experienced men became, they would load up, move to another room and unload in 5 to 8 minutes. The operation is to lift the gathering conveyor with a power hoist, put a pony truck underneath, then the machine is self-propelled, and, being separate on two trucks, it slips around rather sharp curves and narrow quarters without trouble. When it arrives at the face, the gathering conveyor is hoisted, the pony truck rolled to one side and the conveyor lowered and the machine is ready to load coal.

The operation of two Jeffrey 44-C loaders over a period of 223 days resulted in the loading of over 75,000 tons of coal at a supply cost of $782.65 or a fraction over one cent per ton.

In the 3,568 operating hours of the two 44-C loaders at this mine, over 75,000 tons of coal were loaded and only 40 hours lost time because of minor injuries resulted. Under the Indiana compensation law, this means that no compensation was paid for lost time because of injuries.

Speeding Up Rock Work

(*Continued from page* 45)

tractors and operators that the Sullivan and Butler shovels are doing equally as well.

This progress is accomplished by advancing the face two rounds per day the average round measuring 6 to 8 ft. To advance more rapidly or to complete three rounds per day we have found that the Myers Whaley was by far the fastest and most reliable machine. This was proven successfully on the three-mile drainage tunnel of the Glen Alden Coal Co. where for some time we operated on a three-shift basis. However, in this tunnel due to the character of rock (conglomerate) and the necessary time required to drill and shoot we found that although speed per shift was not quite as great, costs on a 2-shift basis were considerably less. This lowering of cost on a two-shift basis is made possible by taking longer cuts and firing the breaking in holes, if necessary, more than once. This additional length of time required for the drilling of long holes and secondary shooting can not be taken on a three-shift basis for, if it is, the overlapping in the shifts causes excessive costs which do not make for economy.

Hand mucking in tunnels is very tedious work and in prosperous times difficult to have accomplished. However, that of mucking slopes, where water must be contended with and rock loaded into cars standing on pitches anywhere from 15 to 60 degrees the condition compared to that of tunnels is far worse. Average advancements of heavy pitching rock slopes hand mucked is hardly more than 4 ft. per drilling and mucking shift whereas if they are mechanically mucked the advancements can easily be increased to 6 or 7 ft. per day. For this work the Sullivan scraper is ideal. This machine has caused a reverse in the grief of this class of rock work. In the past it was always a matter of getting rid of the rock after it was drilled and blasted—today it is a matter of proper shooting due to the fact that on account of the heavy pitch the entire face must be fired at once keeping in mind, of course, that the shots are properly delayed.

Naturally with mechanical mucking equipment one can not contend with anything but proper transportation, drilling equipment and air pressure of at least 80 lbs. at the face. It is useless to try and operate without electric haulage, occasional turnouts (not more than 750 ft. centers) proper ventilation and necessarily good supervision. Besides this bonuses and competition between the different shifts which should be of different nationality, all tend for speed. With this in mind and being sure that the shift boss tends to all details such as seeing that all shots are being prepared for use when the drilling is completed and that all equipment is kept in repair and supplies on hand, the hustle and bustle which the 24-hour job of tunnel driving surely is, will and has easily resulted in the driving of at least 525 ft. of 8 x 12 hard rock tunnel per month.

Anthracite Fines

(*Continued from page* 44)

livered in railroad cars. These unprepared fines are dumped into an outside track hopper and conveyed directly to Rheolaveur after passing over a vibrating screen to remove any coarse foreign material. The feed to the Rheolaveur averages 35 tons per hour.

The coal end from Rheolaveur plant is dewatered by a shaking screen, the water removed going to waste. The tonnage of washed coal recovered per hour averages 23 tons.

The separation from the wash water and subsequent preparation of fines to reduce ash content is, from the point of yield, a wasteful process. A typical example of a plant using a hydroseparator, some ash reducing machine, and classifiers will show on average the following recovery and losses:

Assuming a 600-ton feed of fines in wash water to hydroseparator, about 100 tons will be lost in overflow, 150 tons will be rejected as refuse, and 50 tons lost in classifier overflow, leaving a recovered tonnage of 300 tons, or 50 percent of feed.

SAFETY at the Face

By F. B. Dunbar*

THE subject assigned to me, "Safety at the Face," is so large that a book could be written and not cover the matter. We will attempt in this paper to cover some of the important points and place the responsibility for face accidents.

Mr. Scott Turner, Director of the United States Bureau of Mines, Washington, D. C., states that there were more than 2,000 fatal accidents in the mines during the year 1929, and about the same number in 1930. Of these, more than 1,000 were due to falls of roof and coal at the face.

"Safety at the Face" may be placed under three classifications: *First*, the responsibility of the general management; *Second*, the operating management; and, *Third*, the employe.

In looking over hundreds of accident reports, I find the notation on the bottom of the report in many cases reads something like this: "The accident was caused by carelessness on the part of the injured party." The management is inclined to be satisfied with a report of that kind and will continue to pay high compensation rates, purchase new tires and gasoline for his ambulance in hauling injured men to the hospital.

The general management must assume responsibility for accidents when they fail to give proper consideration in planning the operation. The mine should be laid out so that coal can be produced with a minimum hazard to the face employe.

Many times a system is adopted providing for safe practices and later on the management decides more lump coal is necessary and orders are issued to increase the width of the room and narrow down the size of the pillar. The system has been changed and with it the action of the overlying strata changes and where we had a safe condition under the one system we now have an additional hazard, pillars not strong enough, rooms are too wide and accidents occur. It would have been better to say to the general management that if lump coal is necessary, put in shearing machines or change the method of blasting rather than change the width of the room or pillar. The general management must take the responsibility when they insist upon change in methods.

The operating management is directly responsible for safe practices at the face. Mr. Alex McCanch, one of the senior inspectors in the Pennsylvania bituminous district, states it is well known that "Mankind follows the line of least resistance in all his undertakings and that this inclination is outstanding when men themselves must furnish the motive power." In consequence of this, miners accomplish their task in the most convenient way and with the least amount of labor regardless of hazard. The question, then, of preventing accidents resolves itself into this, namely, How are workmen to be prevented from following their own inclinations regardless of the question of hazard?

"*First:* The obligation is on the management to maintain through their working methods a stable physical state at the face or working place.

"*Second:* Supervision must be intensified both in direction and frequency to the extent that miners are prevented from following improper and hazardous practices.

"*Third:* Discipline must be maintained regardless of cost or production.

"*Fourth:* Intelligent and painstaking effort must be developed in carrying to the miner or other workmen the best and safest practice in performing their various functions at the face."

The mine foremen and assistant foremen, with their helpers, are required by law to carry out certain duties and as the assistants are in direct control of the men at the face, they should be intelligent, conscientious and trustworthy men, who will command the respect of the employes working under their supervision.

It is the duty of the management to hold frequent meetings with all officials, and by all I mean men who are in charge and given responsibility of issuing orders. From these meetings the management can learn who among the officials should receive special instructions and assistance. Educating the officials is as necessary as the education of other workmen.

An assistant foreman, having charge of a section or rib line where 30 to 50 face men are employed, should have as his assistants a fireboss, a coal inspector, at least two shotfirers, necessary timberman, track layers, etc. If this assistant has been instructed properly he will use the help assigned to him and these helpers properly instructed in safe practices with the necessary authority will assist in preventing accidents and the section will produce a maximum tonnage. It is my experience that officials do not give enough thought to the training of men who are in a position to give the assistant foreman proper help and cooperation. I have especially in mind the shotfirer who should be a man of intelligence and years of experience. These men visit the working places two or three times daily and can be of great help to the foremen in maintaining discipline and preventing accidents.

Mr. McCanch is correct in his statement that "Discipline must be maintained regardless of cost or production." We speak of carelessness on the part of the employes injured and overlook the carelessness on the part of the official. Proper investigation may show that the accident was due to insufficient timber or other necessary supplies. Again the accident may have been due to insufficient knowledge on the part of the workman, or insufficient knowledge on the part of the official in charge. A complete investigation of all accidents and a discussion of these accidents by all officials will bring out the necessary information to properly place the responsibility.

The mine management is directly responsible for the employing of all underground men and in many cases sufficient time is not given in instructing the new employes. Frequently the man is placed on pillar work or rib-line without experience, and should have been placed on solid work or with an employe who has had the necessary experience. Frequently when the mine is working part time men are taken from day work and placed on contract work at the face with

* General Superintendent, Mather Collieries, Pickands, Mather & Co.

KEEP SAFE!

Record of **UNSAFE PRACTICES** of.

Check No.________ **Working Place**________________

In this space the various Foremen in charge of, or inspecting the work will note unsafe practices of the workmen and violations of the rules of the Mather Collieries.

DATE	DESCRIPTION OF VIOLATION	OFFICIAL	DATE	DESCRIPTION OF VIOLATION	OFFICIAL

ACCIDENTS, IF ANY________________

RECORD FROM____________ TO____________ 19___

____________ FOREMAN

This is a Protection to

Yourself and Family

Rule 11 of the State Mining law provides "Any person who shall deface, pull down or destroy any notice board, danger signal, general or special rules, record books, or mining laws, shall be prosecuted by the Superintendent, or notice given by the Mine Foreman or obtained from other sources, as provided for in section two of article twenty-six of this act."

The "Keep Safe" card used by Mather Collieries, on which is kept a record of each man's safety violations.

MATHER COLLIERIES **Daily Report of Potential Hazards**

No. Room	No. Entry	Ventilation	Gas	Stoppings Brattice	Loose Roof	Needs Timber	Explosives	Electric Wiring	Haulage Road	Transportation	Fire Hazard	Car Blocking	Clearance	Needs Rock Dust	Electric Machine	Other Hazards			Key—Hazard observed — Ordered removed + Hazard removed ⊕ Remarks

Date________ Section________ Mine Official________

Each foreman carries one of these "Potential Hazard" cards. At the end of the day the mine foreman checks the cards and sees that all hazards either have been removed or the place dangered off.

practically no face experience. Accidents occur and the management is at a loss to understand why. Again we come back to the question of intelligent and painstaking effort on the part of the officials.

In writing to a number of friends regarding this question of "Safety at the Face," one replied, stating that a company had a wonderful system of timbering and specified certain sizes of cap pieces as well as posts. Upon investigating an accident it was found that the company was not furnishing the cap pieces specified in their regulations.

We are required by law to keep every working place safe, and if the miner finds danger at his working place it is his duty to fence off the place and report immediately to the foreman. It is the duty of the miner to sound his roof frequently, and to sound it properly; to block his car, and block his brake; to set his timber according to standards and instructions; to see that all loose slate is taken down promptly; to place his center post or head post, and, finally, he should carry out instructions as to safe practices issued by the foreman or foreman's assistants. Experience teaches us that the face man will not do all the things that he should do, and he will follow his own inclinations regardless of hazard. Therefore, if the assistant foreman or his helpers, find an unsafe condition at the face, or near the face, they should remain there until the hazard has been removed.

We have used a "Keep Safe" card for a number of years. This card is part of the man's equipment and should he fail or neglect to do the things that safety requires he is written up on the card by the foreman and this record is posted monthly and a record is kept of this man's practices. If he continues to be careless and refuses to carry out the safe practices required by the law and the management, he will be discharged.

Mr. George McCaa, of the Pennsylvania Inspection Department of Mines, has prepared a card showing potential hazards. Each foreman carries this card and compares the notes. At the end of the day the mine foreman checks the cards and all hazards must have been removed or the place dangered off.

The rapid introduction of machinery with additional changes has created hazards that we did not have years ago. It now becomes necessary for the machine runner to remove timber and in many cases they neglect to replace it. The machine man is also given an approved safety lamp and the law requires him to use it. A flame-burning lamp in the hands of an official, who knows how to use it, is a good thing. However, when placed in the hands of one who is not familiar with it, it becomes a hazard. A much safer practice is to have a fireboss examine the room before the machine man enters.

A proper inspection and examination of electrical equipment will prevent face accidents. Personally, I would like to see electrical inspectors, employed by the State Department, who would make examination of all electrical equipment.

You will note throughout this paper I have stressed the responsibility of the management as to "Safety at the Face," and this is as it should be. Mining companies that have good accident records are those which have intensive supervision and strict discipline.

Safety with Conveyors

By A. L. Hunt*

THE three methods under which the writer has grouped accidents for the year 1930 are (1) hand loading direct into mine cars; (2) conveyors; (3) scoop loading. The following table gives the total accidents by these methods and the severity rate:

Item	Hand loading	Conveyors	Scoops
Non-compensable accidents	272	137	21
Percent of accidents	63	32	5
Coal mined per non-compensable accident, tons	4,176	5,101	7,999
Compensable accidents	122	60	5
Percent of accidents	64	31	5
Coal mined per compensable accident, tons	9,311	11,648	18,663

On July 7, 1924, experimental work by the Pennsylvania Coal & Coke Corporation commenced with scoops, and shortly afterwards conveyors came on the scene. In time, methods were developed and these two operations became part of our mining system. On December 31, 1930, 3,189,966 tons had been loaded without a fatal accident, one permanent disability, and no partial permanent disabilities.

In the early stages a high percentage of accidents can be attributed to poorly protected conveyor equipment. This has gradually been overcome, until today this equipment is practically fool-proof. Gears and drive-chains have been enclosed, hand grips have been placed on the sides of section pans to enable face conveyors to be readily moved without danger to fingers. Each improvement tends to reduce the number of accidents to a minimum. Some accidents are due to labor turnover. The introduction of new men on conveyor crews is a responsibility; the foreman in charge of this particular work should be careful in his selection, and the instructions as to working cycles should be rigidly adhered to. The writer believes that the most important man on the conveyors is the foreman. He should be mechanically inclined, familiar with conditions at the mine in which conveyors are operating, able to command respect and observance of state and mine safety regulations, without being a bully. He should also have good face bosses on each conveyor, but even then accidents will happen.

We have a rating of 100 per cent in first-aid training; all employes took this course during 1930. Safety organizations are effective at each operation with meetings, each two weeks, to discuss accidents, preventive measures and take any necessary steps to eliminate recurrence. In addition to our local safety organization, there has been instituted through the United States Bureau of Mines, since the first of the year, four chapters of the Holmes Safety Association. These were placed at strategic points to take care of the largest part of our operations, and from personal observation our employes are taking a great interest in this movement. Notwithstanding the fact that more money, time, and effort was spent during 1930, our accident record is not one of which we are proud.

We attribute the fact that conveyors are safer than hand loading in scattered working places, because of concentration and close supervision. The chief hazards to overcome are the dangers arising from dust or explosive gas. A local explosion in a conveyor set up will endanger more workmen than would the same explosion in a room where at the most only two men may be employed. To reduce this hazard to a minimum, all dust is loaded out before shooting and the place is watered. An examination is made by a competent man for explosive gas before shots are fired. The same examination takes place after shooting, in addition to roof test. Where these hazards exist, all conveyor equipment is driven by air and only approved mining machines are used. Another factor for safety is the rapidity with which conveyor rooms are driven and the pillars extracted. Two 40-ft. rooms with a 20-ft. pillar, 330 ft. long, are completed in 26 working days of double shift. In the hand-loading days it would have taken 6 to 9 months to do the same job. In retreating conveyor pillars the day and night crews are subdivided into three smaller crews, this making the operation continuous. Props play a vital part in safeguarding employes. A systematic system of timbering is carried out and nothing but round timber 6 inches in diameter at the small end is used.

We honestly wish to do all in our power to safeguard the lives and health of employes, and we believe that the conveyor will continue further towards reducing accidents in the mines.

* General Superintendent, Pennsylvania Coal & Coke Corporation.

Cost of Mine Accidents

(*Continued from page* 50)

period, or an average of 8 cents on each ton of coal produced.

Corroborating the estimate made of direct cost per ton of compensation insurance, our reports from 28 individual companies, with an annual production of nearly 35,000,000 tons, show the average direct cost per ton to be in excess of 2.5 cents. These costs on tonnage produced range from 1 to 5.7 cents per ton.

It is interesting to observe from these reports from individual companies that the lowest compensation costs are effective among those concerns which maintain well-organized safety departments. One company which mines nearly 2,000,000 tons of coal annually created a safety division in 1928. In that year the cost of compensation was nearly $50,000. In 1930 the cost of compensation insurance to the same company was $16,000.

Another individual company has a direct cost of 4.5 cents per ton. The manager recites that this cost will be reduced to not more than 3 cents during the next year as a result of the reduction of accidents.

Another large company advised that their active accident prevention began in 1928 when their compensation costs were in excess of 5 cents per ton. It dropped in 1929 to 4.6 cents per ton, and in 1930 to 3.6 cents per ton.

The manager writes: "This steadily decreasing compensation is concrete evidence of the dollars and cents value of our accident-prevention work." This same concern has increased its tonnage per lost-time accidents 76 percent in the same period.

"Our present cost of compensation is just a little over $12,000 per year," we were advised by a coal company operating in the Kanawha district. "Five years ago, on the same wage scale and practically the same tonnage, the cost was $24.480 per year."

These instances of individual progress and the general estimate revealing the staggering cost of mine accidents show conclusively that the operator who has studied the cost sheets has become a supporter of accident prevention and has found to his satisfaction that safety pays.

With these astounding figures proving the economic loss to men and companies, and the fact that many companies who have gone into accident prevention with the full intention of eliminating accidents have seen the accident and compensation rate decline, is in itself sufficient reason for concerted action to the end that all preventable accidents will be eliminated.

However, with all the figures showing the monetary losses, no writer or statistician has been able to compute the cost of accidents in heartaches, sorrow, and suffering and oftentimes actual starvation to the mothers, wives, and children who were dependent on the wage earners. In fact, this suffering in many cases is carried on for several generations.

It seems to me that here is a challenge which we must accept, and I venture to say that any company who will set aside a sum equal to one-third of their total yearly compensation costs and organize a safety department and place it in charge of a man who knows accident prevention, who is sincere in his work and can visualize, comprehend and analize conditions that are likely to cause accidents, and who is able to instruct and discipline men, will at the end of three years have cut its compensation costs in half. Furthermore, think of the indirect costs, delays, and suffering that will have been eliminated.

SAFETY With ELECTRICAL EQUIPMENT

By W. P. Vance*

DURING the past few years the use of electricity in the coal mining industry has been closely connected with the introduction of mechanical methods of mining, and the two have been so closely interwoven as to make it almost impossible to present a paper on the subject of "Safety With Electrical Equipment" without to some extent overlapping into the mechanical side of the picture. It is therefore our desire to first ask the indulgence of those on the program who are presenting papers on "Mechanical Mining in Safety With Mechanical Equipment" if at any time during the discussion it might seem that we are somewhat beyond the bounds of electricity in its more closely defined meaning.

This paper will be presented without the usual accompanying tables of statistics, as we believe those familiar with the two sides of the problem as connected with mechanical mining will agree that statistics can be introduced and figures used to prove either side of any controversy arising in connection with safety and the use of electricity or mechanical equipment in our mines.

It has long been a contention of many that an increase in accidents has been inevitable as the use of electricity and the various implements of mechanized mining increased. Many figures have been introduced to show that this is so, and by some the conclusion has been drawn that we must have accidents and an increase in their number in direct proportion to the increase in electrified mechanical operation. Those who subscribe to the belief that accidents increase with mechanization, and must of necessity so increase, lose sight of the fact that the average intelligence of those engaged in the production of coal has also increased as mechanization has increased. Any facts or figures which might be cited as to past experience along the lines of safety with electricity as applied to mechanical operation lose much of their importance when it is considered that they of necessity cover but a short period of time, as it is only recently that mechanized and electrified mining as we have it today has been practiced.

We should, therefore, look more to the planning of our operations in the present and future, and the proper training of our workers in the safe operation of electrical equipment, as we feel that much better results will be obtained in this manner than by proceeding along the lines as outlined by past history. It is our sincere belief after some few years of operation of a very highly mechanized and electrified mine that with the introduction of proper methods of designing and laying out, and proper construction used in wiring and equipping property, and by means of the education of the operators and others in their responsibility for the safe use and maintenance of this equipment, that coal can be mined by these means and accidents almost entirely eliminated if ordinary intelligence is used in their operation.

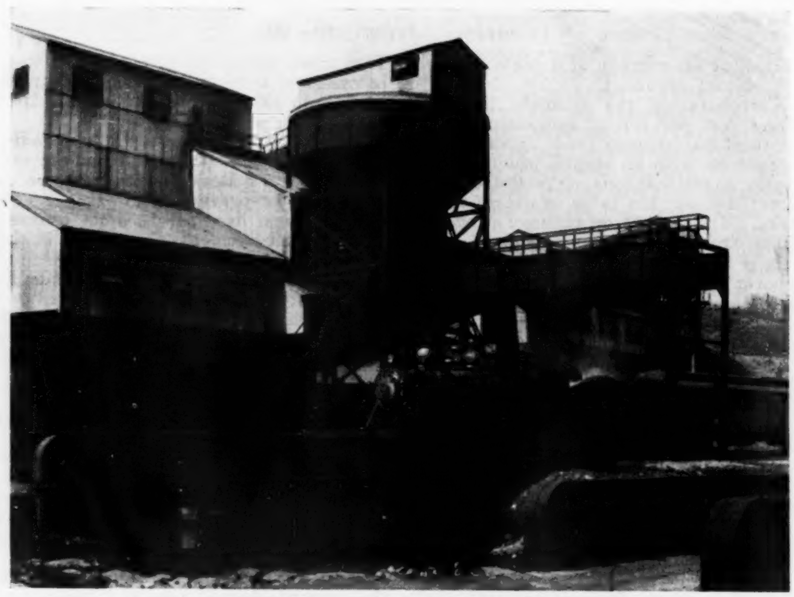
Granted the proper equipment properly installed and maintained, the responsibility for accidents is directly chargeable to the human element, and it thus follows that the education of employes along the lines of safety and safe operation is of the greatest importance. We do not mean by this that we have reached the ultimate in the operation of such equipment, or that we are operating without accidents. It may be stated that such accidents as we have had could with almost no exceptions have been avoided with somewhat more intensive training and education, or by the proper use of the equipment for the purposes for which it was intended, and are in no way chargeable to the use of electricity or mechanical methods of mining.

Safety with the use of electrical equipment must start even before plans are drawn for the use of this equipment, as the first requisite of such safety is a proper appreciation by the executive officers of the company of the dangers involved in the use of large amounts of electrical and mechanical equipment necessary for the mining and preparation of coal. Granted this understanding, it must be kept in mind during the entire planning and designing of the various equipment and operations the one fact that tonnage, preparation and cost must of necessity follow, if the operations are planned to make safety the first consideration, for it is our opinion that no man can function to his best ability having in his mind the thought or fear that the equipment he uses, or the methods in which he uses it, or the general layout of the operations are unsafe. A certain feeling of confidence, well-being and respect for his own ability must necessarily follow if it is the knowledge of the worker that all planning, building and operation of equipment has been done in such a manner as to provide for him and his fellow workers the ultimate in safety.

In our operation every precaution has been taken to install the electrical equipment, making full use of all present knowledge as to the safe installation and operation of this equipment as brought out by the various Federal and state laws and the findings of the various technical societies having to do with mining with electricity. Full use has also been made of the knowledge and latest developments of the various companies producing equipment for use in the mining and preparation of coal. In fact, at all times it is necessary that the manufacturers of the equipment work in close harmony with those who are to install and operate it.

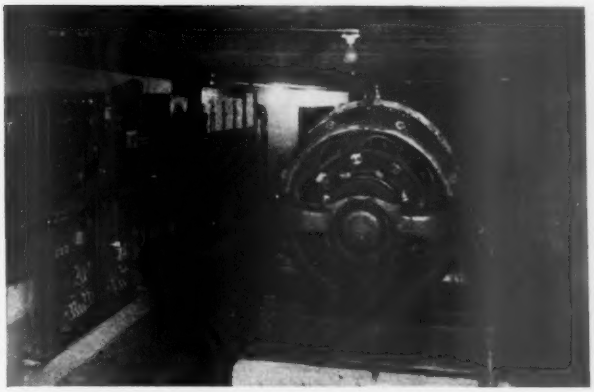
It is our belief that the Safety Department, particularly the safety or compensation rating inspector, should have full knowledge of all plans and should feel free at this stage in the operation to give his recommendations as to the proper methods of construction and operation.

In the past too little thought has been given to the fact that the equipment must be regularly inspected, and maintenance accomplished with as great ease as possible with the result that on many machines used about the mines much more time must be spent in getting to the part for inspection or replacement than in

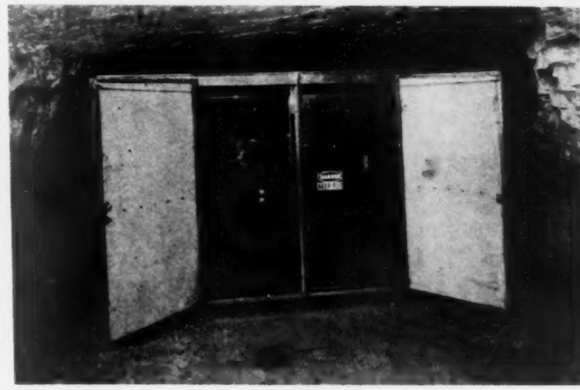


An electrically-operated locomotive fueling station

* General Superintendent, Butler Consolidated Coal Company.



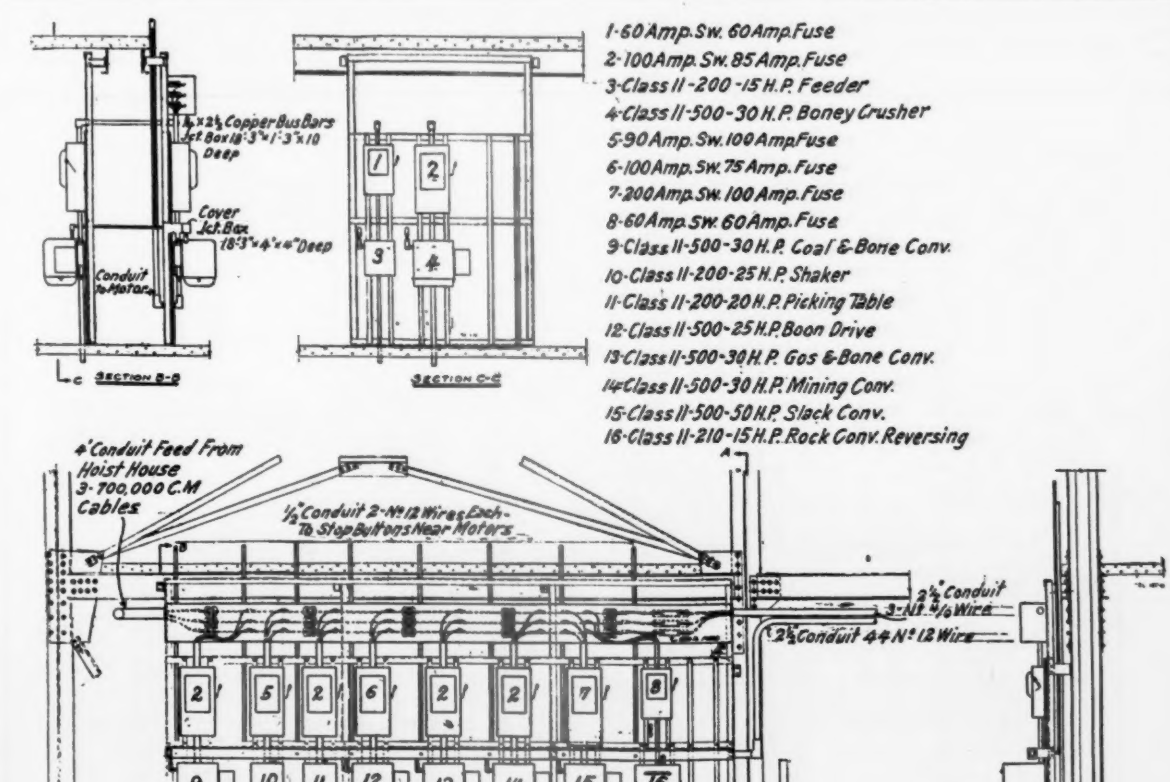
Permanent underground sub-station, housing haulage and storage battery motor generator set



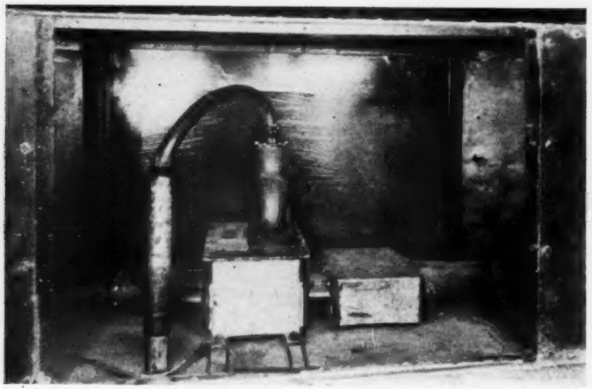
Portable transite generator house

actually doing the work. All contactors, controllers, resistance and other electrical parts should be so mounted as to permit of ready inspection at any time, as these are the very heart of the equipment. In other industrial plants the best of facilities are provided for inspection and maintenance, and the mere fact that a machine or piece of equipment is to be used at or in a coal mine is no justification for installing the control equipment or motors in such inaccessible points as has in many instances been the practice.

We have illustrations accompanying this article illustrating what we believe to be the proper placing of the various control, contactor, and resistance boxes. On the sketch showing the general electrical distribution it will be noted that a great number of automatic and manual circuit breakers are provided between the point where power is delivered to the main station and the points at which it is used in either an AC or direct current motor. These circuit breakers should all be mounted in such a position that proper inspection can be quickly accom-



Distibution system—Typical controller rack



2,300-volt plug station

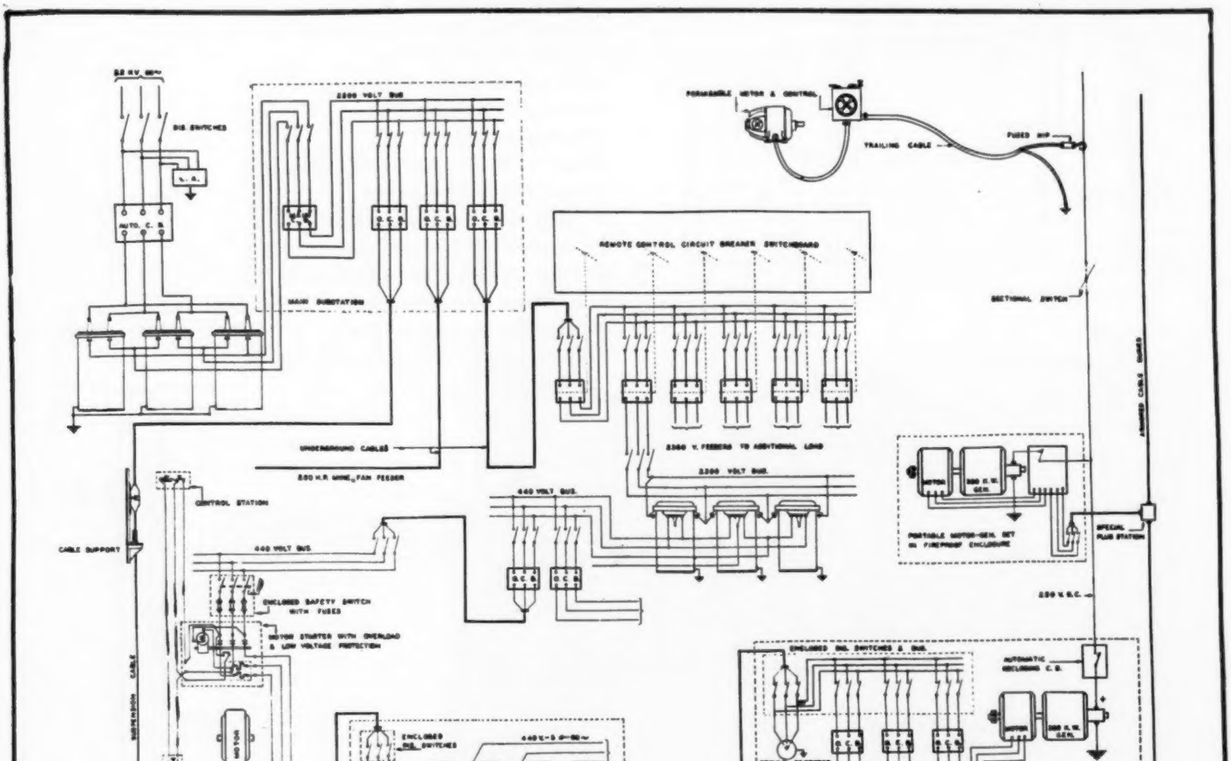
Enclosure for 2,300-volt plug station for portable motor generator set

plished and replacements or adjustments made with a minimum of effort when required.

In the operation of tipples, cleaning plants or other outside equipment at a coal mine we believe that control of the various equipment in each of these operations should be centralized, with indicators at this point to show the operation or non-operation of all the equipment. Switches should be mounted on central control racks, and all connections, including those at the buss bars, should be enclosed.

Safety emergency switches in addition to the automatic control should be provided for each motor. At any points in the operation to which the general public has admittance, or where other persons than the company employes are required to work, either in maintenance or in inspection, even greater caution should be taken that all electrical connections are well protected, as it is often the case that repairmen or inspectors who visit the plant but occasionally are unaware of conditions as they exist, and will take chances which a properly trained employe familiar with the hazards would never think of.

It is a known fact that there are many more accidents during construction work or the installation of new pieces of equipment than during the ordinary operation of the mine. We believe in this instance the mines can do well to follow the example set by many of the large central substations and generating plants of the country where special instructions are issued and precautions are taken during construction periods in an effort to eliminate these additional accidents, which are



Distribution System

often considered to be an inevitable part of every construction program.

If these accidents are allowed to occur at this time it is very possible that the plant in general will gain a reputation as an unsafe plant, and this excess of accidents will have an unfortunate effect upon the morale of the regular employes at the plant, for it is always evident after any serious accident that the men are all working under a more or less great nervous tension and are very apprehensive, and even although all surface indications would show that they are more careful yet it is at times such as this that your accidents usually occur in more or less larger numbers than ordinarily.

It is our belief that the proper place for all power lines and signal lines, both on the surface and underground, is in trenches beneath the surface wherever this is possible. The best obtainable types of submarine armored cable construction have been used with great success at a number of plants. These provide a freedom from atmospheric electrical conditions, and also a freedom from power interruptions in case of fire, explosions, wind storms, or other catastrophes.

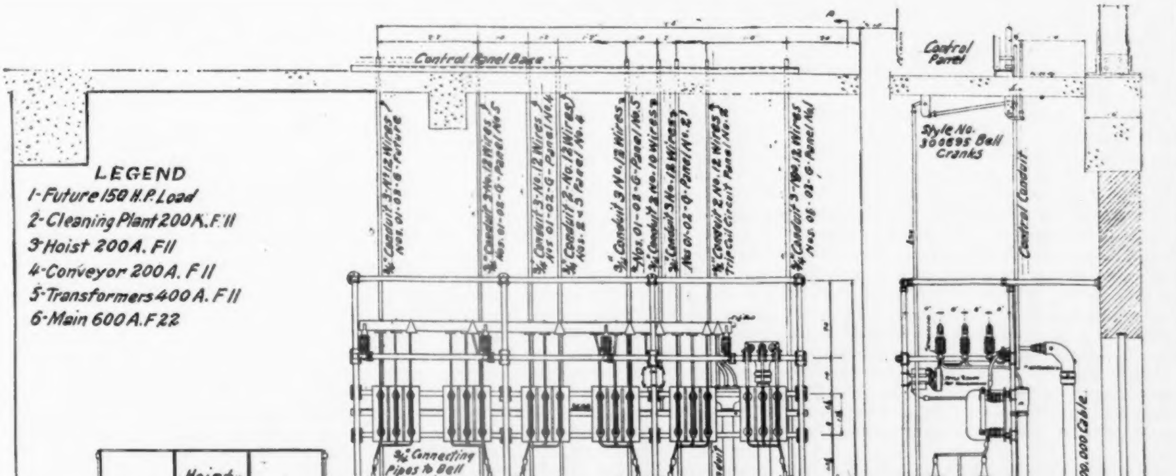
Going on from the proper designing and installation of equipment, we come next to the selection of those who are to operate, inspect and maintain this equipment. Men of a much higher degree of intelligence are necessary for these occupations than for those requiring only a sufficient amount of physical ability to permit the actual work assigned to the operator or laborer to be done. In fact, it is noted that in those operations which are having the greatest success with electrical equipment there are but very few of the old type of miners employed, the men for the most part being mechanics and electricians whose ability is often greater than that of those in the same occupations in other industrial plants. To some certain extent we have found that younger men and men not before employed in mining are more adaptable to the operation and maintenance of such equipment than those who have been employed in the old-fashioned mine with little or no electrical equipment.

We have found it necessary to inquire closely into the education of prospective workmen; also, to have a record as to their former occupations, and if possible the manner in which they performed their duties. Good health and mental alertness are absolutely necessary, as it is evident that if machinery is to be used to its full advantage the operator must be as nearly 100 percent mentally and physically as possible.

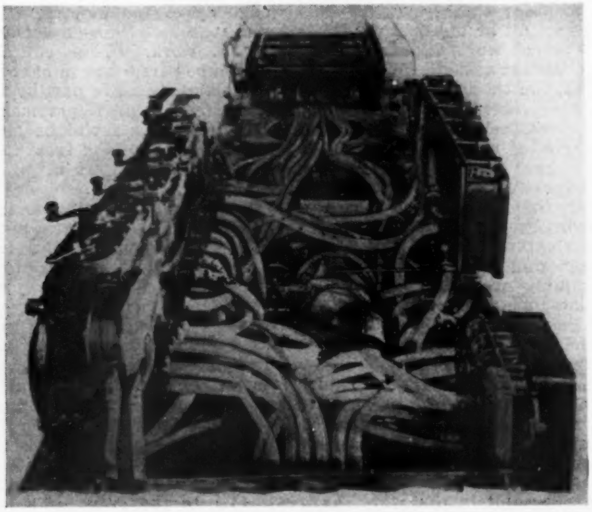
After a man is hired he should be given certain definite training in the duties which he is to perform, and if possible should be put to work for some time on one of the inspection or maintenance crews with other experienced men in order that he may acquire a working knowledge as to the inspection, maintenance and repair of the equipment on which he is to work. This educational work should be performed in part by the Safety Department and in part by the Electrical and Mechanical Departments, as these two departments will be responsible for the results obtained from these men and accidents caused by or to them in the operation.

We have found it necessary after a period of operation to put into effect some very definite rules having to do principally with the maintenance of the various pieces of equipment at the face. It is our belief that rules will differ in various operations and with different kinds of equipment, but that possibly a few general rules will be found applicable to practically all situations. In the belief that these may be of interest to you I will give you some general rules which have been in effect for some time, the proper observance of which has been of material benefit to us in our operation.

(1.) In the event of a breakdown of any machinery in the working faces it will be necessary for the section foreman or a fire boss to be present to test the place before repair work is commenced. Unless the foreman or fire boss is able to remain present until the repairs are completed, he will be required to give the repairman in charge of the work written permission to proceed. In case re-inspection is not made within 30 minutes, the repairmen are hereby instructed to cease



Distribution system—Typical switch circuit breaker rack



Controllers, contactors and resistance mounted in accessible position to facilitate thorough inspection

their work, until again given permission to continue.

In cases where the above rule can not be carried on feasibly, the broken down equipment shall be run or towed back to the trolley wire and repaired there.

Electrical repair work at the face workings should be discontinued as far as is reasonably possible.

The cable nip shall be removed from the line, and pulled back up to the machine while work is being done, and in no case shall it be reconnected to the feed wire until all compartments are enclosed, and sealed.

(2.) No jumpers are to be permitted to be used in the mine at any place. By the word "jumper" is meant, any piece of cable or wire that is placed between the fused nip of any machine cable, and the feed wires as installed by the Electrical Departments. When it becomes impossible to reach a working place with the cable on the machine, the fault may be due to any of the following reasons: Too short a machine cable; lack of wire extension; break-thrus not finished in time to allow feed wire to be extended.

Wire requirements should be figured ahead for at least two days, so that the wiremen can regulate their work and avoid part of the above mentioned trouble.

(3.) No machine shall be operated or connected to the feed wires without having a fuse nip, and the proper fuse in the cable circuit. Machine operators will positively not be allowed to break seals, replace fuses, or make repairs, or adjustments within the permissable compartments of any machine. He may replace fuse or fuse links of the proper size in the cable nips only.

(4.) Each time that a machine cable is spliced, respliced or damaged, it will be necessary for the operator to immediately notify his foreman as to the reasons. The section foreman will then make a report of cable troubles for his shift to the Electrical Department. Cables are to be inspected and checked once a week to confirm the reports.

(5.) When new machine operators are hired, they shall be sent to the machine boss to receive instructions pertaining to the machine they are to operate. This will also apply to operators who are transferred and who have not had sufficient experience on the type of machine they are being transferred to. The Electrical Department will endeavor to provide a demonstrator when required.

(6.) All foremen shall instruct their operators to make immediate reports about any unsafe or wrong conditions of the machines, such as no seals on the compartment covers, broken cables, etc.

(7.) All employes shall be instructed to report all falls of roof or coal to the foreman in charge of section or the mine foreman. It shall be the duty of the mine foreman or his assistant to visit such fall as soon as notified. Any such condition that might exist shall be made safe before leaving or danger boards shall be placed in order to warn any one against entering. In such cases where live electrical wires have been knocked down the Electrical Department shall be notified at once, and it shall be the duty of said department to disconnect the power from such wires at once. It shall be the duty of the person disconnecting power to notify the mine foreman or his assistant when power circuit has been disconnected, and the mine foreman or his assistant shall not leave until such time has been advised.

(8.) On days the mines are not operating it shall be the duty of the mine foreman or night foreman to see that all power circuits are disconnected from all sections of the mine where no pumps or machines are in operation. On sections where power is to be disconnected, the Electrical Department shall be notified in writing by the mine foreman or night foreman. The mine foreman or night foreman shall further advise the Electrical Department in writing when such reconnections for power shall be made, but such notice shall not be given until such time as all power lines have been run, and conditions have been found safe for the connecting of power circuit.

(9.) On days the mines are not in operation, no person or persons shall enter any portion of the mine for the purpose of performing work without permission from the mine foreman or night foreman. Where it is necessary to perform work on electrical equipment or remove electrical equipment from a working face, there must first be an examination made by the mine foreman or one of his assistants. In cases where electrical equipment is taken to motor barn for repairs, and then returned to the section—such machine shall be only taken to the end of trolley, and shall not be taken beyond this point until an examination for gas has been made by the mine foreman or his assistant.

We can not emphasize too strongly that all safety rules and regulations which may be set out for the operation of electrical equipment with safety are only so valuable as the enforcement which follows. Infractions of such rules should be met with prompt and suitable penalties, although we have found that a rule which has been evolved from experience in the operation of the equipment, and the benefits of which have been explained to the operators, is very seldom violated. Education in the dangers resulting from various unsafe practices, and bringing home to the operators the hazard to which they themselves are exposed from such violation, has much to do with the elimination of the hazard.

We have had the cooperation of our local state mine inspector, a man widely experienced not only in the use and operation of mechanical and electrical equipment, but even more widely experienced, and of international reputation, in the results of accidents, explosions and fires caused by the improper use of electricity in the mines, and with his cooperation a system of inspection and elimination of all the potential hazards present in the various operations has been evolved.

Under this system each officer, whether he be fire boss, electrical engineer, assistant mine foreman, or in whatever capacity he may be employed, is placed in the position of a safety inspector. He is given a card upon entering the mine, on which he may note the nature and location of the various potential hazards which he observes. These cards are a written record of the unsafe conditions or practices coming under his observance, and are turned in to the mine foreman or other official whose responsibility it is to see that these hazards are eliminated before they cause actual accidents.

Although this system has not been in operation for any great length of time, we have found that it has eliminated a considerable number of hazards which might well have caused serious accidents, and we feel that when we have become more familiar with the use of this system it will be the means of eliminating the causes of the greater portion of our common accidents, both electrical and otherwise.

As a result of considerable experience with electricity in mechanized mining it is our conclusion that the education of those whose work is connected with electrified mechanization is of the greatest importance, for the human element is present in every operation, from the designing of the plant to the ultimate production of the coal. A careful check should be made that those who design and install the equipment are not only competent to do so, but are keeping before them at all times the various elements affecting the safe operation of the equipment.

Once the equipment is installed it is the problem of the operator to see that all of his employes are properly trained as to the various hazards, the methods by which they may be eliminated, and the proper inspection, repair, and operation

(*Continued on page* 80)

Scrapers and Conveyors
in Thin Seams

By T. F. McCarthy*

THIS discussion, in connection with Scrapers and Conveyors in Thin Seams of Coal, is based on seam thickness ranging from 2 ft. 6 in. to 4 ft., and an average thickness of from 3 ft. to 3 ft. 6 in.

During the year of 1930 there has been a very marked increase in the tonnage loaded by means of scrapers and conveyors. This very radical increase has been brought about by competitive market conditions requiring a quality product at very low prices. The installation of mechanical equipment makes this possible by enabling the operator to secure:

(a) Concentration of working areas, with its resultant reduction in costs of ventilation, pumping, haulage, supervision, mine maintenance and other related mine costs;

(b) Elimination of such non-productive work as brushing in room for car height, track in rooms, and handling of cars to and from the face.

The following are some of the results that may be secured from the installation of this type of equipment:

(a) Increase in production per man. This is possible through the elimination of non-productive work required in ordinary hand methods, in the improved servicing of one mechanical unit over servicing the equivalent tonnage in a number of rooms with hand methods;

(b) In the increased supervision possible through the grouping of large tonnages in concentrated areas;

(c) In the possible reduction of serious accidents, because of the reduced number of men required, and in the elimination of hazards normally encountered in hand methods such as handling of cars;

(d) In the possibilities of better roof action and roof control through the increased speed of recovery.

Scrapers and conveyors are here considered independently and will be described as such. In general, both types of equipment have their peculiar advantages and disadvantages, and the type used will be governed by the characteristics of roof, bottom, and coal seam, market requirements as to quality of product demanded, as well as competitive prices to be met.

Scrapers

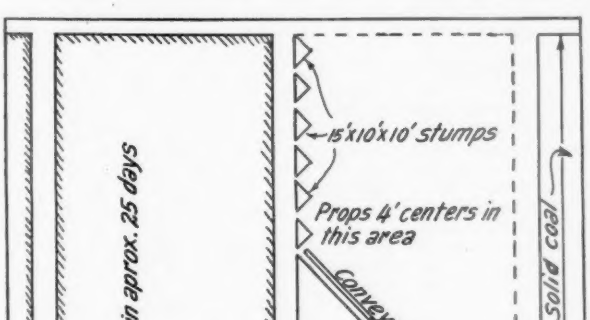
Scrapers may be grouped into two classes: The Stationary Type, and the Self-Contained Portable Type. Both have been extensively used over a period of years, and the results possible from the use of this equipment are generally very well known. In production work the system of mining most generally used is the "V" System, as shown on *Plan No. 3*, or the Block System, as shown on *Plan No. 1*, or a modification of these plans. The Slabbing System is also used with considerable success. Their use in the Standard Room and Pillar System has been somewhat limited, but they have been used extensively in pillar recovery.

Some of the requirements for successful scraper mining are as follows:

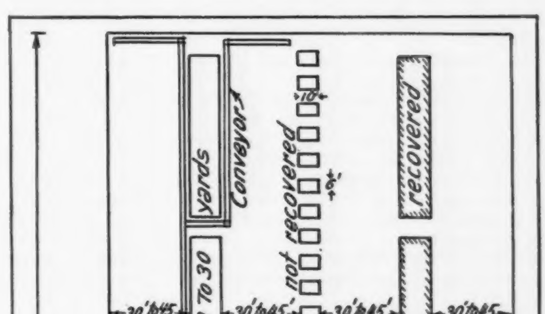
First, a good roof that will permit maintaining the timber line at from 6 to 9 ft. from the face before the coal is shot in order to provide a run-way across the face;

Second, a roof that can be broken on timber in order that a new face will not have to be established when a cave occurs. This requirement is not absolutely essential, however, if the roof does not cave too frequently, as the face can be reestablished very quickly.

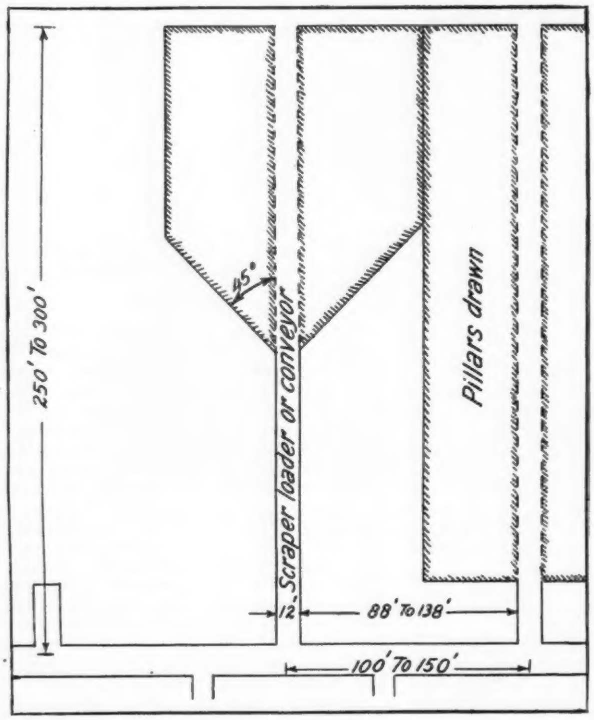
* Assistant General Superintendent, Clearfield Bituminous Coal Corporation.



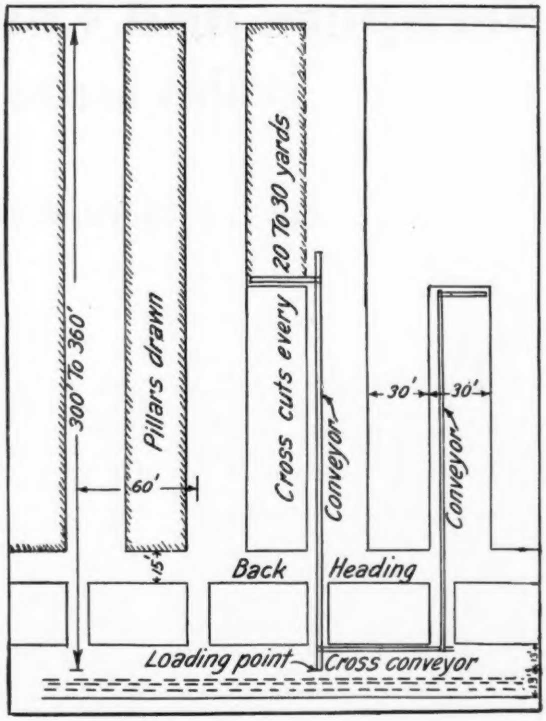
Plan No. 1.–Block System



Plan No. 2.–Double Room System



Plan No. 3.—"V" System



Plan No. 4.—Room and Pillar System

The advantages in the use of this equipment are:

(a) That large tonnages per man employed are possible;

(b) That shoveling can be practically eliminated;

(c) That equipment can be quickly moved when a fall of roof is imminent;

(d) That the equipment can be easily and rapidly moved from one location to another.

The disadvantages in the use of this type of equipment are:

(a) Inability to clean coal at the face, particularly where the seam has impurities in bands other than a free parting band at the top of the seam;

(b) Where the bottom is of a soft nature the scoop tends to dig into the bottom, resulting in impurities being mixed with the coal;

(c) With a soft-natured coal the scoop tends to increase the amount of degradation.

Conveyors

Conveyors may be grouped into three general classes: Chain, Shaker and Belt. All types have their advantages and disadvantages, and the type used is governed by conditions and the preference of the user. Up to the present time the greatest number of installations are of the chain and the shaker types, but the belt type is coming into more general use, particularly as main line gathering units.

There are a number of systems of mining used with conveyors; the one in greatest favor being the room and pillar system with modification as to widths of rooms and pillars. *Plan No. 4* shows the arrangement of equipment when working two rooms with one loading point. The number of rooms may be increased by using a main line gathering unit on the entry.

The Double Room System, which is a modification of the Standard Room and Pillar System, is universally used. This system is shown on *Plan No. 2*, which shows the arrangement of conveyors.

Up until the present time the preference has been for one of the above systems of mining, as they are practicable under almost all roof conditions and are familiar to operating men; also that conveyors are readily adaptable to these systems of mining without any changes in the mine lay-out. The block system, as shown on *Plan No. 3*, is coming into use where roof conditions permit. This long face yields a large tonnage per cut, and increases the ratio of loading time to the non-productive time in a complete cycle, and should yield a larger tonnage per man-day than will the room and pillar systems. The advancing or retreating modified long wall systems are also coming into use with favorable results.

The advantages in the use of conveyors are as follows:

(a) The ability to adapt them to present systems of mining;

(b) Their use will generally increase production per man employed from 50 to 100 percent above that of hand loading;

(c) They can be operated under varying conditions of coal heights and grades;

(d) The relative light weight of the sections makes them easily portable and readily lengthened or shortened;

(e) They permit close timbering to the face and thus afford a maximum of safety in this respect;

(f) Where the coal is shoveled onto the conveyor, cleaning at the face is possible, which is essential to secure a quality product;

(g) The coal can be conveyed from the face to the car without undue breakage;

(h) With the shaking type of conveyor it is possible to use the duckbill or self-loading head. The use of this device prevents thorough cleaning at the face and increases degradation;

(i) In thin seams it is usually desirable to drive rooms to a width of 30 ft. or more and unless the shaking conveyor is carried in the center of room and the width of the room is kept to from 28 to 34 ft. a certain amount of hand shoveling is necessary. It is generally desirable to carry the conveyor along one rib to facilitate the recovery of the pillar, which limits the width of the room to from 20 to 24 ft., and for this reason the use of duckbills is somewhat limited. When using the duckbill it is not possible to carry as close timbering as is possible when hand shoveling is used.

All types of conveyors are adaptable to entry development and their use greatly facilitates this work. The shaking conveyor with the duckbill is especially adapted to this work, as it will load both coal and rock.

As the operator is becoming more familiar with conveyor work he sees the great concentration possible in grouping of units in one entry through the use of a main line gathering unit. Individual units require a man at the loading point, and if top has been taken on the entry extra brushing is required to secure the necessary height to discharge into the car. These two items of cost are greatly reduced as units are combined. The limiting factor, however, is mine car capacity and it is only in mines equipped with large capacity cars that it is practicable to group more than three or four units in one loading point,

(*Continued on page* 66)

LONG-FACE CONVEYOR MINING

Derby No. 3 Mine, Stonega Coke and Coal Company

By J. D. Rogers*

THE Stonega Coke & Coal Company operates mines in Virginia, in Wise and Lee Counties, in what is known as the Southwest Virginia field. Several known workable seams occur in this field, of which five have been worked, or are now under operation by this company.

One seam, which has always been known to be of exceptionally high grade coal, had not been developed to any extent previous to 1930, because of the fact that it would not average more than 39 in. in thickness, and also because it is overlaid 40 ft. above by a high grade merchantable seam known as the "Roda Seam," the thickness of which averages 66 in. In addition to the fact that the upper seam is the thicker, it also naturally follows that from a conservation standpoint the upper seam should be first exhausted.

The quality of the two seams is such that they can be mixed at the tipple without any questions being raised by the customer. From a domestic coal standpoint, the lower or thin seam is of a more splinty nature, therefore will better withstand shipping.

The high coal territory at the Derby Colliery is limited in area, and the time was rapidly approaching when it would be necessary to supplement Roda seam tonnage with that secured from the lower, or "Marker Seam."

Many outcrop openings had been made and coal heights obtained which showed a very uniform seam of coal varying in height from 36 in. to 42 in. of clean coal. Another characteristic which was carefully noted was that in practically all instances the roof was of a solid massive sandstone formation, showing little, if any, horizontal cleavages for a distance of about 35 ft. above the coal. The bottom for approximately 30 in. below the coal is of a shaley fire clay formation, and the next 4 in. is a soft fireclay. This streak is so soft that it can readily be bored with a breast auger.

In making plans for the development of this thin seam, several major points had to be considered:

First—Cost of Production. It was absolutely necessary that the cost of producing this coal should not exceed the cost of production experienced over a period of time in the thicker seam above.

Second—Plans for Development. Owing to the height of this seam and nature of the sandstone roof it was not considered practical to brush roadways and use the standard room and pillar system. The question of the action of the bottom when pillaring should be started was also seriously considered.

Third—Concentration of Production. In low coal mining the necessity for concentration of tonnage from a given area was recognized as most important—both from a supervisory and cost point of view.

Fourth—Safety and Roof Control. Under heavy sandstone cover the question of recovery of the pillars depended largely upon how successfully the roof breaks were handled. The method of handling the roof also most vitally affected the safety of the miners.

Fifth—Selection of Equipment. Since the standard room and pillar method of mining was eliminated, this question, together with choosing of the method of mining to be used, was of the most vital importance.

After carefully considering all the natural conditions, both above and below the seam, it was decided that some method of mechanical mining should be employed, and with that idea in view the Engineering Department was authorized to make inspections of low seam operations having similar natural conditions.

Mining operations in six states specializing or experimenting with low seam mechanical mining were visited and studied with the idea in view of copying or adopting some method to our conditions.

This study extended over a period of more than a year, during which time interviews were had with consulting engineers who had wide experience in low coal operations, and on longface work in particular.

After carefully considering all data assembled, it was decided to secure the major portion of the tonnage from an 800-ft. longface, and reduce the development tonnage to a minimum. Owing to the rolling condition of both top and bottom, 18-in. width belt conveyors which could be quickly extended as the headings advanced were purchased. Cutting was done with bottom cutting machines. The coal was mined and loaded for a distance of 400 ft. or more before bottom rock was lifted in the haulway, after which time the loading point was again moved up and the development continued. No brushing was done in the airways or crosscuts.

It is unnecessary for me to give further details covering the mine development, since there is nothing about it which is difficult, nor does it vary from the experiences usually encountered in driving headings and crosscuts with conveyors.

It is relatively simple to state that an 800-ft. face is to be maintained at all times from which a 6-ft. cut of coal is to be loaded daily, but I wish to assure you that it is no easy task to accomplish. In *Figure 1* I have attempted to illustrate as many details as possible, which have a direct bearing on the production and cost of the coal obtained from the face.

Our study of the problem led us to believe that by means of collapsible cast iron jacks we could hold this 800-ft. face open at all times, and use the system known as "caving longwall." No refuse for packing was available, therefore caving was the only answer to this method of mining. The fact that we were compelled to modify our experiment, and the reasons for the changes, will be discussed later.

First, let me state that the 800-ft. face and headings are really divided into two separate units—400 ft. of the face and the "A" headings comprising one unit, and 400 ft. of the face and the "B" headings comprising another unit. One longwall machine serves each 400 ft. of face, which makes its cut on the night shift. Cutting is followed by men who do the drilling and shooting, so that the coal is ready for the loaders when the day shift starts. Additional men also move up the conveyor to within 36 in. of the face, and, in general, make loading safe for the day coal loaders.

Each 400 ft. of face is tended by a 26-in. belt conveyor—one discharging towards "A" headings, and the other towards "B" headings. Each face conveyor discharges onto a 26-in. belt conveyor (in the aircourse) called a "gate conveyor," and each gate conveyor then discharges onto a 26-in. conveyor opposite the loading points called a "transfer conveyor," from which the coal is discharged into mine cars. The gate conveyor can be extended up to 400 ft. in length, therefore when the aircourse is maintained, loading points need not be closer than approximately 800 ft.

It was estimated that each 400-ft. face would produce approximately 300 tons of coal from a 6-ft. undercut. Due to a very irregular bottom, we found the cutting to be a very difficult operation, and to keep from damaging the product by cutting into the bottom, there was a greater loss of coal than was anticipated. No machine bottoms were scrapped, unless the layer of coal was abnormally thick. The result is that we secure approximately 275 tons of coal from each 400-ft. face, or 550 tons for the total 800-ft. of face.

During the first stages of our experiment the coal loaders were paid by the hour, and it usually took about 26 men to load out the total cut on the 800-ft. face, or a tonnage of approximately 21 tons per man. This was later changed to a price per foot of face, and now we only allow 18 men per face, which means that nine men shovel onto each belt an

* General Manager, Stonega Coke & Coal Company.

average of 30 tons per man. These men seldom take longer than six hours to clean up the entire face.

Figure 1 shows each conveyor in place, and on the "A" heading side the coal is ready for loading, while on the "B" heading side the cut is cleaned up and the face is ready for the night shift, during which time the face is to be cut, and the belt conveyor taken down and set up in the space provided. After the conveyor is moved, the two outer rows of jacks are to be removed so that a fall may be secured and one row of the jacks thus released set up again back of the machine as cutting progresses.

These jacks are set on 4-ft. centers parallel to the face and approximately 30 in. centers perpendicular to the face. The belt conveyor takes up about 42 in., therefore this space must be left clear for the resetting of the conveyor when the cut is cleaned up. After the cut is made and the coal shot down, loading is started. As the coal is removed the fourth row of jacks is set on the day shift, thus completing the cycle of jack setting.

Previous mention has been made of the fact that this seam is overlaid with a massive sandstone about 35 ft. thick directly over the coal, then about 5 ft. of sandy fire clay, which is the bottom for the 5 ft. 6 in. Roda seam. The measures above the Roda seam are practically all sandstone, which are more or less massive. The cover over the block of coal under discussion varies from 250 ft. in the branches to 600 ft. under the ridges.

It might be well to add here that the Roda seam was not merchantable over the area selected for this experiment, therefore the caving longwall method could be used without damage to upper seams.

As the face advanced, all stratas to the surface were either broken, or were bent. This fact was disclosed by profiles being run parallel to the face, and levels checked at intervals of time. No cracks on the surface were found, but a general settlement averaging 14 in. was observed over the major portion of the excavated area.

The cycle of operation above described worked out perfectly until the face had advanced 200 ft. from the barrier without a fall of any kind, and absolutely no weight showing on the few wooden props which were set for observation purposes. To induce a fall, a row of holes was drilled in the roof about 4 ft. in depth vertically, and the entire row shot at one time. This cut a trench in the roof the entire length of the face, and about 135 ft. from the coal barrier. Still there was no fall worthy of note. This, of course, weakened the roof, and as the face continued to advance falls began to occur. It was not until over 300 ft. of area had been opened up that a general fall occurred which, we think, went above this 35 ft. of sandstone directly above the coal.

It might be interesting to note that while I was at last year's meeting of the Mining Congress I was advised of this major fall. No damage was done to the conveying equipment, and except for some small falls between the conveyor and the face, the jacks held the face open. Several jacks were broken, and a few were never recovered on account of the fact that they were pushed down into the bottom by the excessive weight of the monolithic blocks of sandstone. Loading of coal was delayed two days, after which time the usual cycle of operation was maintained until usually after every third or fourth cut we would again have a major fall with the resulting delays and a loss of jacks by crushing. These major falls, however, were not as violent as the one resulting from the original break.

It was our hope and idea that as more area was exhausted, that we would secure daily breaks along the jack line which would relieve the pressure along the face, and prevent interruption in the loading of coal. If breaks could be secured along the line of the outer jacks, the roof over the conveyor and loading space would be maintained in a safe condition and extra timbering eliminated. The sandstone was so rigid and massive that when the weight came on the jacks, the jack settlement was great enough to make the break line come at the face of the coal, and frequently about 3 ft. beyond the coal face.

In other words, about every third cut a fall would be secured. Instead of this fall breaking off along the outer row of jacks, it would usually break along the working face of the coal and absolutely shatter the roof directly over the loading face and conveyor; the jacks, however, would hold the face open, and only occasionally allow minor falls of roof in the open spaces along the face and over the conveyor. These could soon be cleaned up with slight expense and work resumed if they occurred during the work shift. The real damage was later reflected in the labor cost to move the jacks and conveyor when making ready for the next cut and day shift. Roof was broken up, and frequently several jacks were crushed; many others were forced down into the bottom and were most difficult to recover. The danger to workmen was also great, therefore it was often necessary to use many wood timbers to recover the jacks.

The face was kept in operation in this way until it was demonstrated that mining conditions had probably been determined which would remain fairly constant, and if that be true, then a modification of the scheme was necessary. It was costing too much money on the night shift to prepare for loading on the day shift.

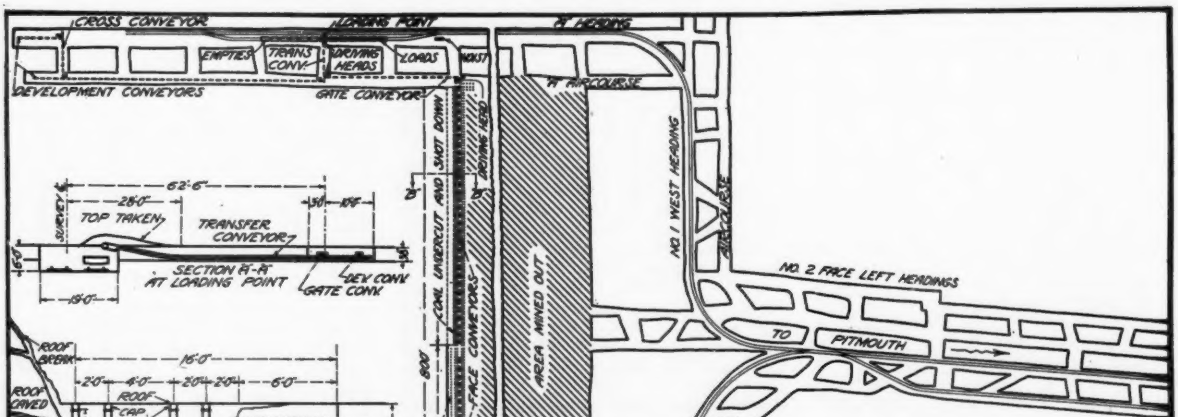


Figure 1

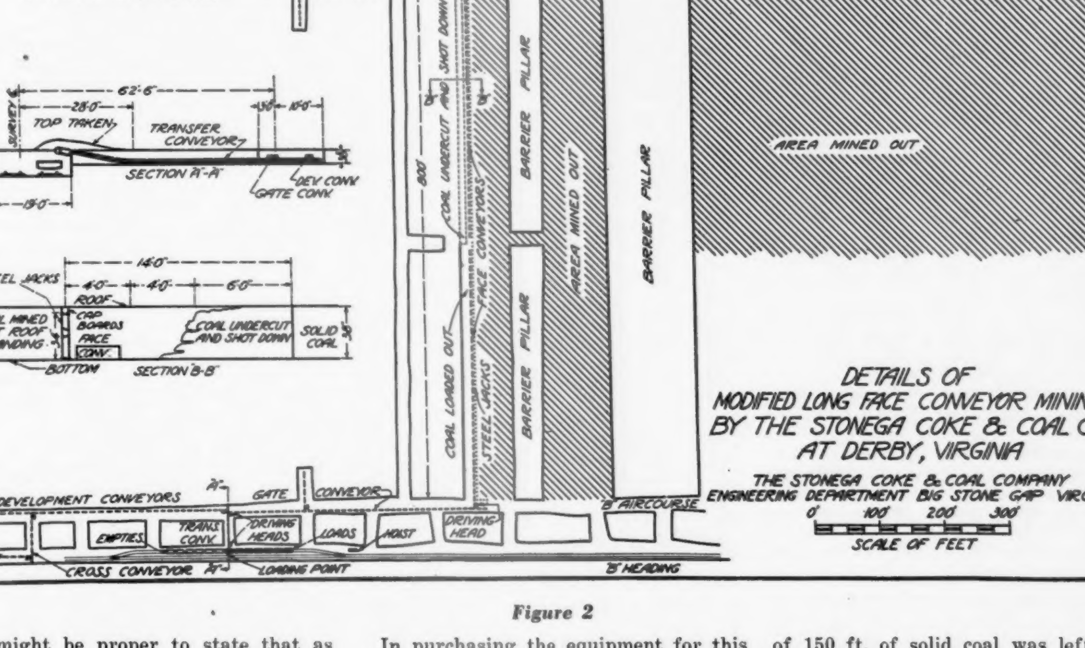


Figure 2

It might be proper to state that as shown in *Figure 3*, this face was advanced 1,050 ft. before it was decided to change the method of mining. The tonnage was fairly dependable, the average being slightly better than five cuts per week; but other conditions led us to believe that we must capitalize on this good roof if we were to realize our fixed cost of production figure. The roof was too good and too strong to lend itself to this method of mining. Had our roof conditions approached several installations visited, which have successfully used a method similar to the one we tried to use, I have no doubt but that we would be still following out our original plans. Engineeringly speaking, the plan secured 100 percent recovery, and was successful, but from a coal cost standpoint it was not satisfactory.

In purchasing the equipment for this installation it was recognized that the scheme described above was a rather extensive experiment; therefore, whatever was purchased might have to be used with some other method, as modification of our plans might become necessary.

When this became apparent, it was decided to eliminate the idea of 100 percent recovery, and leave the necessary pillars to support the roof. Experience had taught us that we could reasonably expect no falls in a space of 150 ft. x 800 ft. The next point to decide was how much coal it was necessary to leave and then mine another open space of 150 ft. x 800 ft. General information led us to believe that about 30 percent of the coal would be sufficient. In order that no weight might be thrown over onto the solid coal from the old excavated area, a barrier of 150 ft. of solid coal was left and a room was driven from each side. The face conveyors were then removed from the old face and set up in the room and slabbing started. All jacks were then removed from the old wall and the roof allowed to cave up to the face of the coal. Inspection reveals that this occurred, and the falls have closed the entire face. Coal is loaded by hand onto the conveyor and conveyed to the loading points in exactly the same manner as previously, but when 105 ft. of space was excavated, the conveyors were again moved into a new room driven in advance, leaving a 50-ft. pillar to support the roof.

It is thought that a space 150 ft. wide might be safely slabbed under this roof, but for safety reasons the maximum distance allowed was 105 ft.

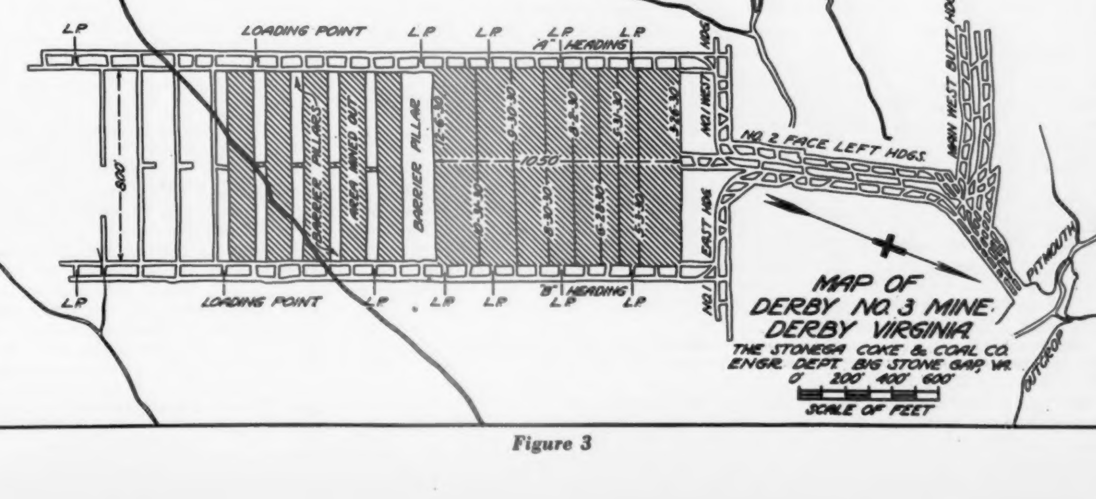


Figure 3

Even though the roof was excellent, a row of jacks was set on 6-ft. centers just outside of the conveyor, and were moved up with the conveyor. No wood timbers were used except where there appeared a sandstone scale or "horse back."

The moving of equipment has been greatly simplified, as the conveyor is now skidded or slipped over after loading is finished, and is never dismantled until it is set up in a new room. The crosscut shown in the 50-ft. barrier pillar was made principally to facilitate the moving of the conveyor parts and jacks when the new set-up is made. No machine bottoms are scrapped unless rolls in the bottom cause the cut to be made so high in the coal that it is deemed advisable to prepare the bottom for the next cut.

It might be interesting to note the number of men used on each 400-ft. face each day and night, and from this determine the tons of coal produced per man working. (See tabulation.)

Figure 3 shows the development of the mine up to date as of March 1, with a total of four rooms slabbed and two other additional rooms driven and ready for the installation of the face conveyor when needed. As of May 1 nine rooms have been slabbed and abandoned. At present writing we can see no reason why this 800-ft. panel can not be completed as projected.

To illustrate the extreme strength of

	Day	Night
Foreman	½	...
Loaders	9	...
Car trimmer	1	...
Conveyor engine tender	1	...
Cleaners along face belt	2	...
Drillers and shooters	2	...
Timberman	1	...
Machine men	....	2
Conveyor movers	....	4
Jack men	....	2
Foreman	....	½
	16½	8½
Total men employed per cycle		25
Tons produced per 400-ft. face cut		275
Tons loaded in mine cars per man		10.6

the roof over this seam, I will state that in the nine rooms completed to date, there has been only one fall worthy of mention. In No. 3 room a large slab of sandstone, probably 30 in. in thickness, has fallen near the center of the panel. There are no indications of weight showing in these rooms, and the remaining pillars do not appear to be under heavy pressure.

When the present panel reaches the limit of the property, it is proposed to retreat with a 400-ft. face on the right of "A" headings and on the left of the "B" headings in exactly the same manner. There seems to be no apparent reason why this retreating coal can not be mined more cheaply than that taken from the present advancing panel. All brushing is done, main line tracks and sidetracks are laid, loading points established, and all main power circuits completed.

Mention has previously been made regarding the condition of the pillars left, and that no weight of importance is yet showing. We have yet to experience roof action during the retreating of a panel on either side of the one now advancing. We are not yet prepared to say whether we are leaving too much or too little coal in the pillars.

The Stonega Coke & Coal Company has a very large acreage of this low coal, and we are really using this mine as an experimental one. We are not yet satisfied that we are operating this seam in the most satisfactory and economical manner. It is a fact that we are now considering the possibility of using our 26-in. conveyors as "mother" conveyors, onto which we will discharge coal from wide rooms—say, 70 ft. wide—on room conveyors, supplemented at the room face by a 60-ft. face conveyor. This scheme has wonderful possibilities, and if paper figures are any guide, the coal should be put on the car much more cheaply than is being done at the present time. The whole experiment has been most interesting, and we look forward to the coming year's work, confident that we will be able to eliminate some of the difficult features of the proposition which are yet giving us cause for worry.

Scrapers and Conveyors in Thin Seams

(*Continued from page* 62)

as with small capacity cars it would require too frequent car change.

One operator in a thin seam has projected, and is now puting into operation, a concentrated plan of mining in which there will ultimately be installed 9-room conveyors, working four units from each side of the main line gathering unit. The extra unit will be available for new set-ups in order that eight units will be in continuous production. The room conveyors will be of the chain type, the main line gathering unit will be composed of three 500-ft. belt type units. The entries will be developed with shaking conveyors, using duckbills. As this mine is equipped with 4-ton capacity mine cars, this plan is entirely feasible.

A summary of a representative time study of a double room system, as shown on *Plan No. 2*, is given below. The equipment is composed of one main line unit with auxiliary and face conveyors. Each room has its own undercutting shortwall machine and electric coal drill. The crew is composed of three face men in each room and one man at the loading point. Cutting machines are equipped with 6 ft. 6 in. cutter bars. Rooms are approximately 45 ft. wide—coal height 3 ft. 2 in. Production per man averaged 15 tons per man, including the man at the entry.

From this summary it is seen that only 40.4 percent of the time represents actual production work; the remainder of the time being consumed in nonproductive work incidental to loading, or in delays that can be classified as avoidable and unavoidable.

Our problem is to work out methods whereby the non-productive time in the cycle may be reduced and productive time increased.

SUMMARY

Item	Man minutes	Percent	Remarks
Start	5	0.1	
Loading	1,432	40.4	Includes slack from cuttings.
Cutting	313	8.8	
Drilling	131	3.7	
Making shots	34	1.0	
Tamping	56	1.6	
Moving conveyors	100	2.8	Includes extension of conveyors
Timbering	120	3.4	
Cleaning undercut	166	4.7	Before shooting.
Changing bits on machine	50	1.4	
Shooting	92	2.6	
Waiting on cars	166	4.7	
Eating	127	3.6	
Idle	48	2.8	
Repairs	259	7.3	
Greasing and oiling	98	1.4	
Moving machine	12	0.3	Moving into cutting position and out of way.
Power off	35	1.0	
Miscellaneous	298	8.4	
Total	3,542	100.00	

As mechanical mining brings into use new types of equipment, modification of old established methods of mining and changed labor conditions, there are many new problems to be solved.

Probably the most difficult problem is in the attitude of labor to the new order. Under old methods, the miner was his own boss, his working periods were intermittent and he had no mechanical equipment with which to contend. In our new methods, he must work steadily and with machinery. He must be trained and educated in using and caring for this equipment.

The installation of equipment changes the duties of the supervisory force. It usually permits of more intensive supervision, but brings with it the necessity of supervision of mechanical equipment which carries with it the necessity of a knowledge of such equipment.

It brings with it a problem of maintenance that becomes important if production is to be maintained. Repairmen must be trained in the care of new types of equipment. Their problem becomes one of prevention rather than that of repair.

It brings with it new problems in Safety and Accident Prevention. Particular care is required in the wiring of equipment to reduce hazards. Guarding of equipment is important and must not be neglected.

Mechanical mining with scrapers and conveyors is the salvation of mines operating thin seams, and is here to stay. The many new problems presented in this method of mining are far from being solved satisfactorily, but their solution is possible and the operators in thin seams are working on them, as are the "Tasty Yeast Jestors," with "vim and vigor."

CONVEYORS in THIN SEAM MINING

By C. C. Hagenbuch*

IN considering the problem as to which type of loading equipment is best adapted for a particular mine or coal seam, the choice lies between (1) mobile track equipment, (2) pit-car loaders, (3) scrapers, (4) shaker conveyors, (5) belt conveyors, and (6) drag conveyors.

In determining the type of equipment to be used, we must make our selection after having studied the following major factors:

Grades.—Adverse grades, while permitting other types of loading equipment to work successfully, may prevent the success of a shaker conveyor installation.

Thickness of Seam.—The thickness of seam governs our choice of equipment, as we could not expect to select a machine requiring 5 ft. of head room and use same in a 3-ft. seam where otherwise it would be unnecessary either to brush top or to lift bottom.

Nature of Pavement.—Soft or scaley bottoms work against the use of any type of digging loader, as well as against shaker conveyors and scraper loaders. Digging loaders and scraper loaders scale the bottom, mix same with the coal, and cause preparation difficulties. Shaker conveyor drives will frequently work themselves loose on soft bottom, destroying the conveyor alinement and creating maintenance problems.

Nature of Roof.—With any type of loading machine, it is policy to secure the maximum possible tonnage per move per set-up. This must be done by providing either a wide or a deep cut, or a combination of both. If the nature of the roof is such that increasing the width of cut requires the use of crossbars, or of an excessive number of posts, then the cost of timber, labor, and delays to loading will reduce the savings otherwise made by the installation of mechanical coal-loading equipment.

Mining Systems.—It is extremely improbable that a mining layout which has proved to be suitable for hand loading will prove adaptable without change for machine loading. If hand loading has proved what roof, bottom, and rib action may be expected under certain conditions, it is then advisable to consider a loading machine that possesses adaptability to a system as closely related as possible to the hand-loading system which has proved to be successful. It is essential that the system selected should provide: (a) Safety to the workmen; (b) a maximum amount of coal per set-up; (c) a short distance between loading faces; (d) quick car-changing facilities; (e) nearby side tracks; and (f) ability to control the roof and pavement.

Gassy or Nongassy Mines—All mechanical loading equipment can be used in nongassy mines, but for use in gassy mines only those types of permissible equipment should be considered which bear the approval plate of the U. S. Bureau of Mines.

Amount of Fine Dust Which May Be Stirred Into Suspension.—In coal beds where handling creates suspended dust, it is advisable, in order to decrease the explosion hazard, to adopt a type of mechanical loading equipment which will permit the minimum amount of stirring, throwing, dropping, or agitation of coal while being loaded.

Impurity Bands.—An impurity band close to the bottom of the seam may, if not too hard, be cut out by a short wall or long wall mining machine, and the coal loaded out by either mobile or immobile loading machines. However, if the impurity band is located so high in the bed that undercutting machines can not reach it, and if it is advisable to remove this band by cutting, then track cutting machines and the employment of mobile loaders is advisable.

Effect on Size of Product.—Naturally, excessive digging, agitation, and dropping causes excessive degradation. Where size is important, careful consideration should be given to the amount of handling necessary with different types of equipment.

Impurity Extraction at Face.—Where the seam contains free impurities which can not be removed by the mining machines, it is important that those men working at the face have an opportunity to hand pick the impurities. Little opportunity is presented for this important function by that class of machines which we might say "shovel their own coal." Where hand loaders shovel the coal onto the loading machine, face preparation equivalent to the ordinary hand loading preparation may be obtained.

Structure of Coal.—In some few cases where coal is extremely hard or lumpy, and is mined in large blocks, selection of the proper loading or conveying equipment is quite difficult. Such coals usually possess enhanced market value, due to size. It is important that equipment be selected that will handle these large size blocks, and do so with minimum degradation.

Size of Mine Car.—When conveyor or scraper loaders are to be used, the size of the mine car is not of paramount importance, as cars are spotted at the loading point in trips and car changes are made without loading interruption. However, when pit-car loaders or track loading equipment is being considered, then the matter of car changing has an important effect on the capacity of the loading machine. Car changing represents one of the largest lost-time items in the operation of these types of loading machines, and it is quite evident that a 3-ton car, as compared with a 1½-ton car, will reduce the car-changing losses by 50 percent.

Cutting Machines Available.—As a rule, mechanical loading equipment is installed without the purchase of new type cutting machines. When bottom-cutting machines, which can be operated off the track, are available, then no restriction is placed upon the type of loading machine selected. When track-cutting machines only are available, then conveyors or scraper loaders should not be selected.

Maintenance Cost.—Probably the most important item affecting the savings possible by the installation of mechanical coal loading equipment is that of maintenance cost. Whenever possible, data should be secured relative to upkeep cost of various types of equipment working under conditions similar to those prevailing at the mine where installation is proposed.

Effect of Breakdowns on Output.—In this case, we have the old question of "Is it better to have all the eggs in one basket, or several?" Naturally, a high-tonnage machine with several motors and intricate mechanism contains a greater breakdown hazard than a lower tonnage machine with one motor and a simple mechanical layout. Suppose that we consider the purchase of a large machine which will load 300 tons per shift, and that for the same sum we can buy three machines, each capable of producing 100 tons per shift, and also that for the same expenditure we can purchase 10 machines, each of which will produce 40 tons per shift. In case of a breakdown with the 300-ton machine, we lose the total output, whereas a breakdown with one of the smaller capacity machines only causes the loss of a portion of the output.

Rate of Advance Possible.—When mechanical loading is adopted for development purposes, it is important to select a type of machine that will give the maximum advance in narrow work. The development possibility of the machine selected is affected by seam height, distance between working faces, nature of roof, and other features.

Tonnage Increase per Man.—It is almost invariably true that mechanical coal-loading installations are considered with the primary idea of securing an increased output per man. Therefore, the possible percentage of machine loaded tonnage increase over hand loading for various types of machines installable should be given careful consideration.

Possible Reduction in Loading Rates.—Increased output per man will permit scale reductions. A careful investigation

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and study of possible reductions should be made, together with selection of mode of payment as between day rate, straight contract, or contract and bonus.

Organization. — Mechanical loading equipment will not produce desired results simply by installation and the selection of a mining system. Supervision must be intense, as companies can no longer countenance superficial observance of processes by foremen. Complete success can only be assured by the whole-hearted cooperation, constant supervision, and keen initiative of the entire organization.

Cost Credits and Debits.—Before selecting the type of mechanical loading machine, a clear idea of just where cost savings may be expected and of where additional expense may be incurred should be had. The savings, as well as the added expenditures, are not always tangible. The main items where cost decrease may be accomplished are:

A. A decreased loading wage due to increased tonnage per man.

B. A decreased gathering cost where mine cars are spotted in trips instead of individually.

C. A saving in yardage in seams where small height prevents the placing of mine cars in the rooms without brushing.

D. Savings in material and labor due to elimination of switches and track where scrapers or conveyors are used.

E. Savings in gathering haulage, main-line haulage, and rock disposal when brushing of top or bottom is eliminated.

F. Savings in track, wire drainage, and ventilation due to decreased territory necessary to be opened to secure the same tonnage.

G. Timber and tie savings. Due to rapid extraction of coal and completion of panels, it is seldom necessary to replace ties or timbers on account of deterioration, the original installation serving throughout the life of the panel.

H. Savings in supervision and maintenance due to concentration of tonnage.

I. Capital expenditure savings for house plant due to increased tonnage per worker.

J. Increased sales in times of good market, due to quick tonnage increase in shorter period than possible with hand loading.

Against the savings just enumerated, we must consider the following as debit charges for mechanical loading equipment:

A. Increased power costs due to added demand and energy used.

B. Increased depreciation.

C. Added loading machine maintenance cost.

D. In cases of conveyors and scraper loaders, cost of moving and setting up.

E. When hand cleaning can not be accomplished at the face, increased cost of tipple picking.

F. Under certain conditions timber requirements in excess of hand loading.

Cases have occurred where mechanical installations have been made and later abandoned. In some cases this has been the result of improper machine selection, as for instance: Shaker conveyor installations where coal must be moved up excessive grades. Machine installations with misfit mining systems or installation of machines insufficiently rugged to withstand handling of hard-structure coal. However, the fact that mechanization is meeting the coal industry's need for decreased costs is proved by the rapid growth of mechanization installations.

The Consolidation Coal Company operates several mines in which the major portion of the tonnage is loaded mechanically. These mines are in different states and operate different seams. One operation has been mining coal mechanically since July, 1928, working a 34-in. coal bed. Two others, which I am going to discuss more fully, started in February, 1930, and September, 1930.

Having a background of considerable experience from our first operation, we were well aware of the correct answers to most of the questions arising in the consideration of mechanization of the second mine. The following physical conditions were common to this seam of coal, and were mainly instrumental in our choice of conveyors as the proper type of mechanical equipment:

1. Grades running as high as 17 percent.
2. Seam thickness of 28 to 34 in.
3. Soft bottoms.
4. No impurity bands.
5. A roof, seam and pavement broken, but not slip faulted, due to the previous mining of a 10-ft. coal bed approximately 120 ft. below.
6. Ability to work a more concentrated system of mining, due to freedom from any concentrated roof pressures.
7. Ability to increase the percentage of lump coal and larger sizes due to change from a pick-mining system to wide-room work with mining machine undercutting.
8. Saving of 16 percent of the former cost per ton by elimination of yardage in all but main haulageways.

The small coal height and possibility of yardage saving confined our choice to some method of conveyor mining. The far too steep grades and soft bottom conditions eliminated shaker conveyors, and an experiment of several years ago with longwall mining, which had failed, due to the broken nature of the top rock, was responsible for the discarding of belt conveyor equipment as a possibility. The choice then centered on a drag conveyor installation and wide-room mining.

Past experience with wide-room mining had suggested the advisability of securing a greater tonnage from one heading than that possible of being produced by the operation of one or two wide-face rooms. We therefore developed a plan of working simultaneously four rooms on one butt heading; two rooms off the aircourse side, and two rooms off the entry side. In making this layout, we took advantage of the broken top conditions, and of the fact that we could not expect any considerable amount of weight to carry over and cause crushing or squeezing. In passing, it might be added that several butt headings have been completely mined out, and that the concentrated room system has proved quite successful.

Former hand mining methods required the lifting of bottom to provide mine car or locomotive clearances. The cost saving in rooms due to elimination of yardage was so attractive that decision was made to further this advantage, place conveyors in the butt entry, and eliminate all yardage work except that necessary on the main haulageway and the side track at the mouth of the butt entry. Accordingly, our main entries are spaced at such an interval as to leave a 1,200-ft. solid block of coal. A butt or room entry is driven by drag conveyors 600 ft. into this block, the half block on the right side of the main heading being mined out as the main heading advances. When this right half block has been mined to the limit of the main heading, operation is changed to the half block on the left side of the main heading, and this 600 ft. of depth mined with the retreat of the main heading. After the room entry has reached its limit, room conveyors are placed on either side, driving four rooms, each 40 ft. wide, for a distance of 300 ft., and the entry block mined on the retreat. A 20-ft. pillar is left between rooms, and is drawn after the rooms have reached their limit. A 10-ft. pillar is held between every pair of rooms and is never recovered.

The opposing pairs of rooms are staggered just sufficiently to permit the room conveyors to feed into the entry conveyor without interference. The entry conveyor consists of two 300-ft. drag conveyors in series, feeding to mine cars on the side track, which is parallel to the main haulway.

Cars are placed in trips and changes made by either car retarder or slow-speed hoist, without stoppage of the conveyor. Choice as between car retarder and slow-speed hoist is dependent upon the amount of grade. One man is always stationed at the loading point, from which he controls the operation of the entry conveyor, cars, and distribution of supplies.

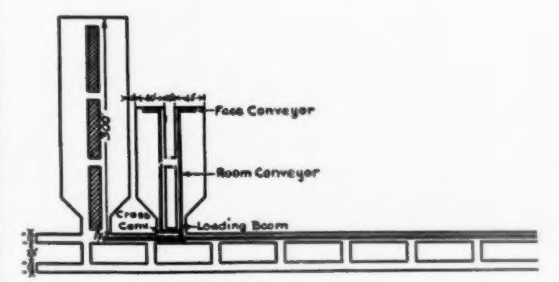
Considering that the maximum travel of 900 ft. of 34-in. or lower coal was more than we could ask of our employes, we have installed a 24-in. gauge track of 12-lb. rail on steel ties in the entry only, and built buggies for use of the men in traveling these entries. These buggies are on 5-in. ball-bearing wheels and permit the men to lie flat and work themselves up or down the entries without fatigue. Each buggy is equipped with brakes. The track, because of its light weight, can be installed or removed in sections, and represents a very inexpensive arrangement.

Power is conducted to the top of the conveyor entry by a two-conductor positive cable with a bare cable serving as a return. Each of the positive conductors has its individual braker and switch located at the boom operator's position on the side track. One of the positive conductors serves all the conveyors, the second serves the mining machines, blowers, and electric drills. By this arrangement the conveyor circuit may be cut off and power still be available for the blowers and other machinery. Each conveyor drive unit can be controlled by an individual switch located at the drive unit.

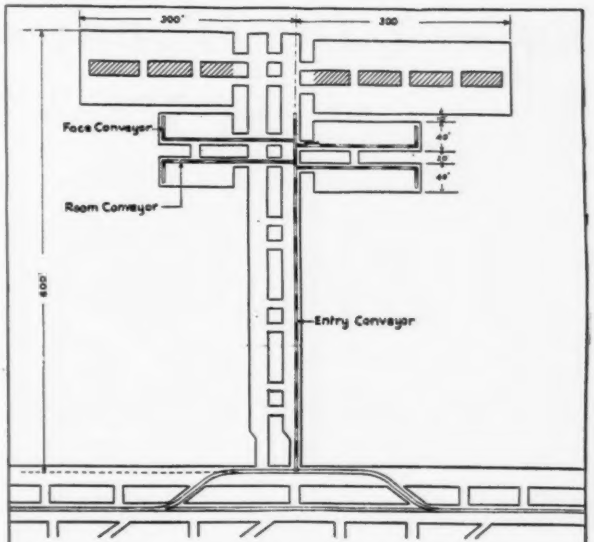
Two telephones are installed, one at the mouth, the other at the top of the entry. The telephone circuit is connected to the regular mine system and conversation can be held from the conveyor room mouths to the outside.

Supplies are handled by reversing the conveyors. They are taken in when a fresh crew goes on shift. Two men are stationed at the 300-ft. mark, where the heading conveyors meet, to transfer the supplies from the first to the second heading conveyor. Other men are stationed at the mouths of the rooms, where the supplies are unloaded and stored until required at the face.

Sufficient conveyor equipment has been installed to operate a butt entry with four rooms working and in the meantime complete driving of the butt entry next above. We have not experienced any difficulty in timing, having in every instance completed the driving of the new entry



TWIN ROOM SET UP
DRAG CONVEYORS
WITH
SINGLE LOADING POINT



QUADRUPLE ROOM SET UP
OF
DRAG CONVEYORS
FEEDING TO ENTRY CONVEYORS
SINGLE LOADING POINT

before the rooms were worked out on the entry below.

It is worth mentioning that our mining system requires just four main heading track turnouts—two for the side track to the room operating entry, and two for the side track to the development entry. The four turnouts are used and reused, and the number will not need to be increased until we install additional conveyor equipment.

Where breaks, as the disturbed strata condition is termed, are encountered, we are naturally slowed up in our progress with a consequent reduction in tonnage output per man. Fortunately, breaks rarely occur in all faces at the same time, and though a tonnage decrease occurs in one portion of a mechanical section, output from the balance of the section is normal.

We believe that a contract wage system is necessary to create production incentive, and have applied this principle to all mechanical operations. However, for some time we could not determine the correct incentive method applicable to this installation, our trouble being the uncertain physical conditions due to encountering breaks, and the necessity of familiarizing our men with the new mining system. For these reasons we paid our conveyor workmen on an hourly basis. Lately, however, we have installed a wage system predicated upon a guaranteed base hourly rate, with a bonus per hour for each one-quarter ton mined in excess of a base daily tonnage per man. This method insures an adequate rate of compensation when physical conditions prevent an average loading, and encourages effort for extra tonnage when loading conditions are normal.

As has been previously stated, the yardage saving on every ton of mechanically mined coal effected a cost reduction of 16 percent as compared to cost secured under hand-loading conditions. Fifty-seven percent of the tonnage is conveyor loaded and the yardage saving applies to only this 57 percent of the output. The saving on this total output due to elimination of yardage is therefore but 9 percent. However, since the conveyor installation has been made, we have doubled our tonnage and made a cost reduction of 17 percent on the entire output.

The Consolidation Coal Company operates a third mine in which drag conveyors produce a large portion of the tonnage. This mine works a distinctly different seam from the mine already described, said seam being 46 in. in thickness and gaseous. Due to this gaseous condition and the rigid ventilation restrictions imposed by the state mining law, we can not attempt the four-room concentrated system already described, and have effected no greater concentration than the working of one twin-room unit on a room heading.

Conveyor operation of twin rooms is by no means new. Several years ago this company tried such a system, working a main conveyor in one room and an auxiliary conveyor in the second room, same being connected by a cross conveyor placed in a crosscut and moved toward the face whenever a new crosscut was driven. This system had several disadvantages, the most important of which were: (1) Two points where supplies had to be transferred from one conveyor to another, these points being at both ends of the cross conveyor; (2) operation of the main conveyor whenever coal was loaded at either face; (3) necessity of stopping the auxiliary conveyor at times when coal was being loaded from the auxiliary room face, before extension of the main conveyor was possible. Because of these disadvantages, use of the twin-room system was abandoned. However, when conveyor mining was inaugurated at this mine in September, 1930, a method of working twin rooms was adopted which eliminated all of the disadvantages enumerated above. The small sketch submitted with this paper shows the manner in which this was accomplished.

A 300-ft. drag conveyor is installed in each room, only one conveyor possessing a loading boom. The heading length between the two room necks is widened out an additional 6 ft., making the section of entry at the mouth of the twin room 18 ft. wide by 40 ft. long. Within this offset we install a cross conveyor serving to transfer the coal from the auxiliary conveyor to the loading boom at the mouth of the main room. The tail end of the loading boom is extended 3 ft. horizontally, the flat space taking the coal served by the cross conveyor. The loading boom is of sufficient capacity to handle the coal when both faces are loading. Supplies are placed on the conveyor at the mouth of each room and the conveyor reversed to move the supplies to the face.

Room, pillar, and centering dimensions are the same as those already mentioned for the 2½-ft. seam of coal.

All face equipment, mining machines, drills and face conveyors are permissible.

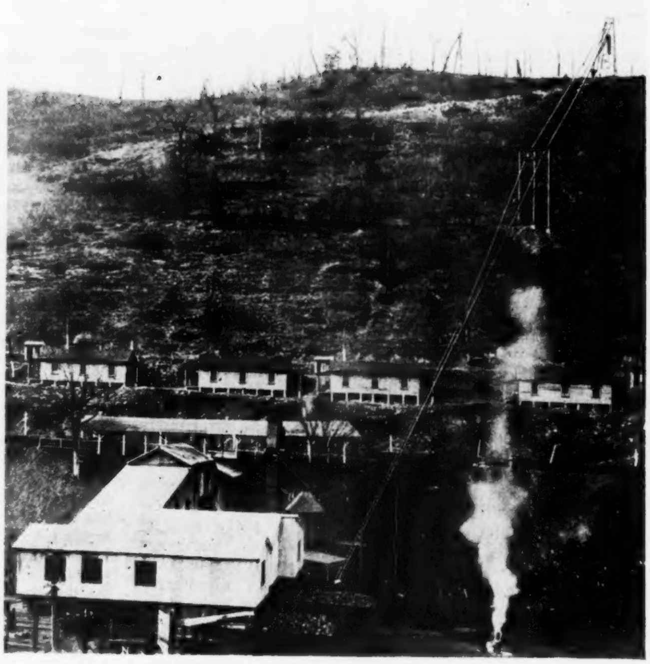
Each set-up of twin rooms is worked by a single crew of nine men. One leader-cutter, one cutter, one boom man, and six crew members. Four men are assigned to each face. Payment is made on a contract basis at a set amount per ton, each man sharing equally in the tonnage produced. The leader and the cutter each receive a small amount per ton in excess of the other crew members. The crew provides their own explosives.

The average tonnage per eight-hour shift for all men employed is between 14 and 15 tons per man, this figure including the moving of conveyors between set-ups. On individual shifts, we have frequently produced in excess of 19 tons per man, or 170 tons for one crew.

Successful Handling of Mine Refuse

By F. S. Follansbee*

To the right is the tail tower where waste is spilled down a steep hillside



Double reversible type slate disposal tramway of the Octavia J. Coal Mining Company, at McAndrews, Kentucky

IN MOST coal mines there is a certain amount of refuse that it is more economical to bring to the surface than to gob along the rib or haul to abandoned workings.

Where production is limited through lack of haulage equipment, hoisting capacity, tipple or washer capacity, or some governing factor that restricts the flow of coal from the face to the railroad car, it is sometimes better to handle the waste material on the off shift. This, however, usually requires additional day men.

Where handling refuse during the production shift does not cause loss of production, nor interfere with the routine underground or surface work, the same day men who handle the coal will handle the refuse up through a certain point. This point should, if possible, be the dump, where one crew will dump both coal and slate, the separation being made by a flop gate underneath the dump, or, if this is not possible, by two dumps side by side, in the case of an end dump car, or two dumps in line, in the case of a rotary dump car.

Where this arrangement can not be worked and it is necessary to dump the refuse cars at some other point, it may be possible to shunt them onto a side track below the coal dump, from which point they must be handled by the surface refuse crew.

After passing through the flop gate under the dump, or onto the side track below the dump, cost of handling and the method used depend on the location of the tipple with respect to the point of final disposal and with respect to the topography of the local surface.

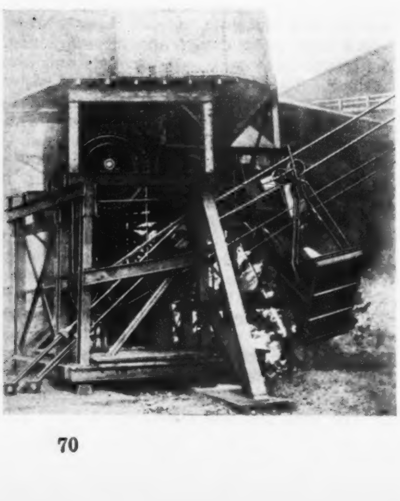
A drift mine in a mountain side usually presents the simplest problem of refuse disposal. Here, if the outcrop has sufficient elevation above the main valley, a side ravine may usually be found large enough to last the life of the mine. In this case, if the edge of the spur between the ravine and the main valley is within one or two hundred feet of the headhouse, a drag chain in a trough will take refuse from directly under the coal dump. This drag may be lengthened from time to time, and if there is trouble from slides at the discharge end of the trough, the last section may be in the form of a boom held in place by a mast and cable. The boom may be swung in an arc providing for a large volume of disposal between extensions of the main conveyor.

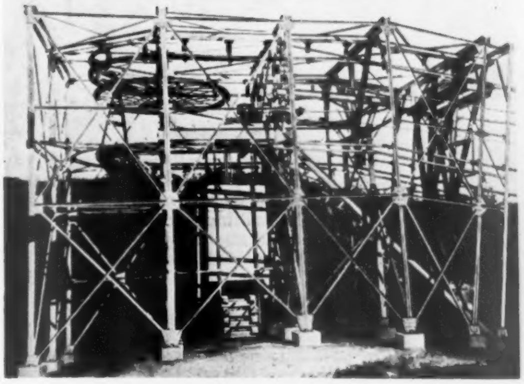
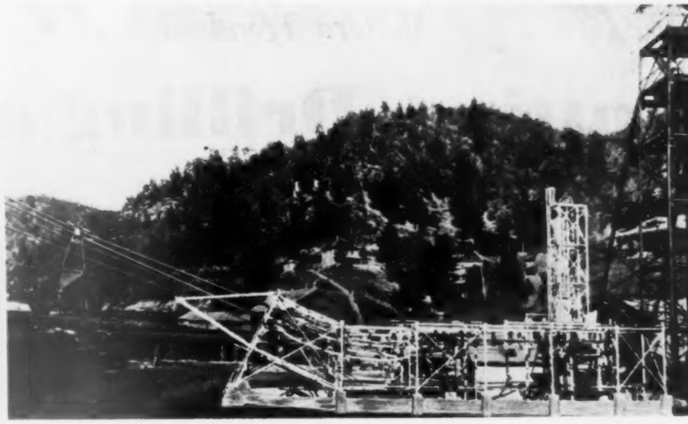
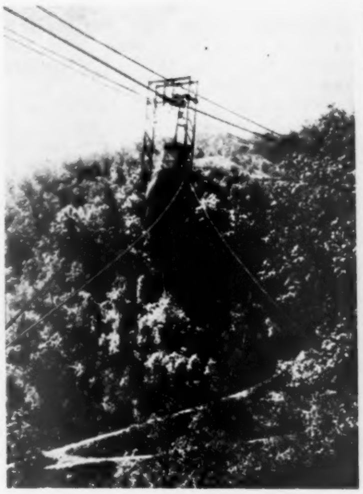
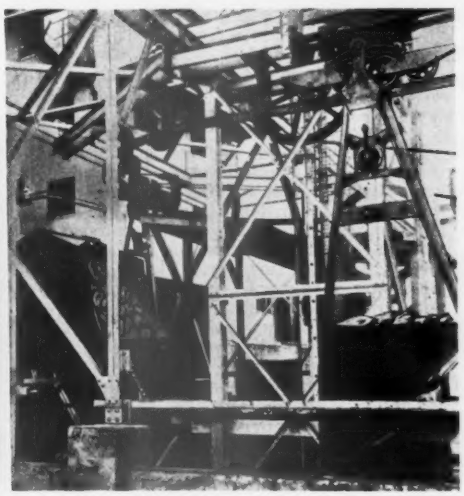
Another method is a small storage bin under the coal dump and either a motor-driven larry along the hillside or a self-dumping refuse car pulled by a hoist up grade to the dumping point and returning by gravity to the bin.

A satisfactory disposal, if the distance to the ravine is not too great, is a one or two bucket aerial tram. On a span of from one to two thousand feet, several years of disposal may be had, depending on the amount of refuse to be handled and the depth of the ravine. With this arrangement the bucket dumps when the direction of travel is reversed

* Chief Engineer, The Koppers Coal Company, Inc.

Below—The loading terminal



Four views of the tramway of the Houston Collieries Company at Kimball, W. Va. Capacity, 75 tons per hour.

Upper Left—The tail terminal. 1" and 1⅜" locked coil cables; ¾" traction rope. Upper right—The loading terminal.

Lower left—34 cu. ft. end dump bucket, discharging at tripping frame. Lower right—Apron feeder conveyor in loading terminal.

and the dumping point may be selected at the will of the operator, using a short haul when there is a heavy run of refuse from the mine and a long haul when the run is light.

Where refuse cars can not be dumped over the coal dump and must be shunted onto a refuse track, they may be hauled to the dumping point and discharged over any of the several types of side hill refuse dumps.

These dumps turn the car over sidewise as in a rotary dump, and, if used with an end dump car, the end gate may be held down by an attachment on the dump.

The next type of mine is where there is a shaft or drift in the valley.

In this case refuse usually must be carried up into a side ravine or, in some cases, across the ridge into the next valley.

To get into the side ravine requires larrys climbing, say, a 6 percent grade, refuse cars hoisted on a slope, an aerial tram, a conveyor, or a combination of these.

To cross into the next valley the simplest solution is usually an aerial tram.

The third type of mine is a shaft or slope in level country.

The usual method here, if the refuse can be stored close to the dump, is a refuse car and hoist on a 30 degree slope, taking refuse from a bin at the tipple.

The advantage of the aerial tram over the larry, hoist or side hill dump is that it eliminates track work, trouble from slides and from burning refuse dumps. One man can operate an aerial tram, where, with other methods, more labor is usually required.

The first cost of refuse disposal plants will vary with conditions. A trough and drag chain 100 ft. long may cost $1,500. A side hill dump from $3,000 to $4,000, depending on the size of the mine car. To this cost must be added the cost of the track to get to it.

A 10-ton larry will cost about $5,000, plus the track it uses.

A refuse car with hoist about the same.

A two-bucket aerial tram with a 1,700-ft. span, rope speed of 1,200 ft. per minute, capacity 50 tons per hour, on an upgrade of 15 degrees, using a 40-hp. motor, $19,000.

A single-bucket tram with a 100-hp. motor for the above conditions, $16,000.

A continuous bucket, automatic dispatching aerial tram, with a 100-hp. motor, making a 1,000-ft. span from the loading terminal, on a 25 degree slope to the top of the ridge, and a 1,600-ft. span across the valley to the top of the next ridge, capacity 75 tons per hour, $50,000.

Some rope manufacturers have guaranteed cables for 600,000 tons of refuse. This, of course, will depend on the length of time required to handle this tonnage.

The cost of operation of the continuous bucket tram mentioned above, written off in 10 years, is, for maintenance, depreciation, interest, power and operation, about 11 cents per ton of refuse, if it is run at its full capacity for six hours out of the eight-hour shift, or 450 tons daily.

This aerial tram takes refuse from a 300-ton bin, into which is dumped ashes from the boiler house, refuse from the washer and straight mine refuse from the hoisting shaft. When the capacity of the mine increases to the extent that the aerial tram can not handle the refuse during the eight-hour shift, the 300-ton storage bin will allow its capacity to be doubled by running a second shift at the extra cost of one man plus the power. The life of the disposal area, without shifting the tail tower, is 10 years.

In any successful refuse disposal scheme the most important part is to cut the amount to be handled to a minimum.

Modern Trends in

Cutting, Drilling and Blasting

By G. C. McFadden*

SINCE engaging in mechanical loading of coal practice has continually changed as a result of experimenting in individual rooms and in entire sections so that gradually methods have been adopted which seem to be the most practical.

The objective in cutting, drilling and blasting is primarily to produce the maximum amount of lump coal with mechanical loading. This is limited by the desirability of securing the largest tonnage per loader. Hence there is a continual search for a compromise to produce lump coal and at same time permit maximum production.

CUTTING

In mechanical mining the breast machine has practically passed out of the picture, and indications are that the shortwall machine is rapidly becoming obsolete.

The track mounted machine has many advantages and some disadvantages as compared to the shortwall. The original machines installed have proven successful, but with the constant endeavor to do the work better and cheaper we have recently installed a machine of the track mounted type which has incorporated a number of novel features which have considerably increased the speed of handling of the machine, and consequently increased the production per machine.

It is desirable in a track mounted machine to carry the minimum number of tools and accessories. Two jack pipes, one keg of sharp bits and one keg of dull bits and an oil can are all that are carried and handled on this machine. It will be seen that these accessories require very little time in loading and unloading.

After the machine is run up to the face it is desirable that the cutter bar may be swung quickly to the cutting position; likewise when the cutting is completed time is saved by the quick swing to the straight position as the tramming motor pulls the machine from the face.

* Assistant Vice President, Peabody Coal Company.

A track mounted machine cutting its maximum width cuts a crescent shaped kerf tapering from nothing at the edges to the maximum depth straight ahead; hence a desirable feature incorporated in this machine is four changes of feed or swing of the cutter bar. This enables maximum feed to be used in starting and finishing the cut and the other intermediate speeds as limited by the hardness of the cutting toward the center. This feature saves time by enabling the feed to be adjusted properly according to the depth of cut and hardness of cutting.

Power driven lifting, lowering and tilting of the cutter bar enables changes in level of the cutting to be made without slowing down the cutter chain or the feed. In some sections the track has local dips which make it desirable that sand boxes be added to these track mounted machines to prevent stalling in the tramming operation. With the above features incorporated in a track mounted machine about the only time lost is in setting bits. The next paper to be presented this afternoon deals on the important subject of bits and no doubt will cover such items as time lost in setting, power lost account of running with dull bits, effect on the grade of bug dust with dull and sharp bits.

The position of the cut naturally has a primary bearing on the drilling and shooting of the coal. At first the machines in one group of mines were cutting on or near the bottom in order to provide a smooth surface for the loading machine to operate on. However, in that position the cutting was the hardest of any portion of the seam; also cutting on the bottom left a serious problem of cleaning a band from the coal. The location of the cutting was then changed to a point directly underneath the band that was in fairly good coal and relieved the cutting somewhat by that change. One cut was made underneath the band and then a second cut through the band, depending on the bits to pull the bands out of the kerf. This resulted in an unavoidable mixture of the dirty band with the screenings and other fine coal.

The latest method has been to cut above a second small band called the steel band. It can readily be seen that cutting in the middle of the seam complicates the drilling and shooting by the fact that approximately half the coal has to be shot up and the other half shot down. This proved to be a rather difficult problem to solve but after continued experiments the routine of this work has been perfected so that good results are now obtained.

In this practice all band is loaded out with the coal and cleaned in the tipple.

DRILLING

The drilling problem has been serious, not so much as to the drilling itself, but in securing the holes at the right place to produce the best results.

In order to shoot up the bottom half of the coal to leave a smooth bottom, it was found desirable to drill one row of holes so that the back of the hole would be on the bottom or even penetrating slightly into the fire clay. At this location the drills often struck boulders or sulphur streaks very difficult to cut through.

We are now using the heaviest drill motor and post obtainable and while these parts stand up reasonably well the trouble is transmitted to the drill heads and cutters.

Even at this time further tests are being made with drill heads and cutters to reduce the cost of sharpening and to cut longer without changing. Cutters tipped with hardened surfaces have been tried. The four pointed molefoot has probably given as good satisfaction as any but are still not entirely satisfactory.

To facilitate the drilling operation light push cars are used to transport the drills and all accessories from place to place.

The cutting and drilling operations lead up to the final problem of blasting, but it is a combination of the three with the many variations possible which affect the final results.

With the mining machine cutting in the middle of the seam, the lower layer of holes, four to six in number depending on the width of the room, are drilled with the two outside holes gripping slightly into the solid so that these shots will bring out the corners and square up the places. The two inner shots are spaced approximately half way between the center and the two side ribs of the room. The four shots required to bring down the top coal are spaced somewhat similar to the bottom holes but usually down about one-third of the way from the roof to the machine out.

The order of shooting is two center lower holes, then the two outside lower holes, then the two center top holes, finally the two outside top holes. This is general practice; however each mine as well as each section of the mine requires some variation of the above system to give good results.

In all these operations close supervision is of prime importance to secure the most favorable results.

Treating MACHINE BITS

By H. H. Taylor, Jr.*

SINCE the advent of cutting machines in mining practice there have been many improvements in the machines themselves and in the methods of applying them. Many of the mechanical improvements have probably been the direct result of experience on the part of the mine management or the machine operators themselves. Because of the lack of facilities at the average mine, most of the improvements in design and material used have been worked out in the well-equipped shops of the various machine manufacturers. With a few minor exceptions, the only part of the equipment which has been left to the mining companies to buy direct, on their own specifications, has been the bit steel. This part of the equipment may be purchased from many sources, while the other important parts, like the machines themselves, are manufactured by comparatively few concerns.

In order to make this discussion of value and interest, we must satisfactorily answer two questions:

Can undercutting costs be reduced by the use of improved machine bits?

If so, does the additional expense of providing the improved bits counteract the saving effected through their use?

Any individual considering this problem would have to answer these questions for himself by applying to his own conditions the factors involved.

Importance of Cutting Machine Bits

When bit steel arrives at the mine it is, in a sense, a raw material, and the process of turning out a finished bit for use underground is under the supervision of the mining company alone. It is peculiarly significant that this part of the cutting unit requires more time and labor to maintain and replace than any other.

Bits affect the cost of cutting directly and indirectly in many ways. Direct additions to the cost of cutting attributable to bit steel alone may be as follows:

(a) Cost of bit steel. (b) Cost of heating and sharpening. (c) Distributing to machines.

There are other factors in cutting costs which may also be affected by the bits but which are not dependent on them alone, such as:

(d) Changing bits at machines. (e) Power consumption. (f) Machine repairs.

The personal equation, as introduced by the care of the individual machine operator and the shop crew, the adequacy of the power and the natural cutting conditions, enter strongly into the picture in these factors.

Coal is generally undercut on the tonnage basis or on the day basis. Under either system the first three factors (a, b, and c) and the last two (e and f) are likely to be the same, but the factor "d" deserves special mention, as it depends upon the system used. Under a tonnage system such as is generally followed in hand mining, bits are changed on the machine runner's time, and he suffers a loss in potential earnings. Under a day scale the company pays for this lost time. Every delay chargeable to bit failure raises the cost of cutting in either case by reducing the amount of coal the machine crew can cut during the shift.

All the factors mentioned above have been at work for years, but it is only recently that we have begun to realize how much coal a machine could cut for us if we would eliminate costly delays and allow the speeding up of the cutting action by giving the machine some good, sharp bits with which to work.

It is, of course, recognized that cutting conditions differ and that a given machine with a certain chain and a certain type of bit will cut coal much more cheaply in some seams than in others, and that consequently the advantages to be gained by improved bits will be more attractive from a "dollars saved" standpoint to a mine with hard cutting than to a mine with easy cutting. However, there are few coal operators who have not tried something to improve cutting efficiency at one time or another. To obtain this end through improvement of the design of machine or chain, we must depend upon those capable of working out the complications of such designs in manufacturing plants. However, there are no mines where the simple operations involved in bit preparation may not be practiced by the most inexperienced.

At this point it would be well to state that, as far as is known, bits were first prepared by simple hammer and anvil methods, later with hammer on a stock and die, still later on a mechanical trip hammer, and finally on specially designed bit-sharpening machines. It is assumed herein that all bits prepared in mine shops today are on either mechanical hammers or special machine sharpeners.

The importance of sharp bits of lasting quality was forcibly impressed upon our company when we began to seriously consider machine capacities. Resistance to abrasion was the quality most desired. Heat generated by friction was known to make bits red hot in any kind of hard cutting, and there was no doubt that an ordinary steel would wear away faster in a heated condition. Therefore, any substance offering more resistance to abrasion than ordinary bit steel and remaining hard while at a red heat should be an improvement. As we saw the problem, it would be necessary for us to improve the quality of our bit steel, improve our methods of sharpening and conditioning the bits, or apply some foreign substance to the bit points harder than the steel itself.

Methods of Improving the Bits

Of these three, we considered the bit steel itself first. It was a standard product sold to many mines and handled by any number of houses, being .75 to .85 carbon, .50 to .60 manganese, less than .04 P and less than .04 S. We were told by competent metallurgists that a higher carbon steel would give better results if properly forged and heat treated, but, because of the high carbon content, this steel would be more easily burned in the forging operation, and, with the crude methods used at the average mine, good results would not be as readily obtained as with an even lower carbon steel. We made inquiries about the possibility of getting a steel more resistant to the cutting action and found that many such steels were available, but that most of them were expensive alloys which could cut plenty of coal but were far too high in first cost to be considered. Most of the alloys used for special tool work are hard and tough, but when the points become dull, as they would in time, the simple method of applying an ordinary forging heat and resharpening would not work to satisfaction on any of these alloy steels. It was found that expensive and elaborate heat-treating equipment and highly skilled labor would be necessary, all of which helped us to forget special steels for the time being.

Before passing up this alloy possibility entirely, we considered the application of such a steel cut into very small bits.

If the bits could be small enough, it would seem that a high-priced steel might be used and the bits discarded after becoming too dull, thus eliminating the necessity of special heat-treating apparatus and, in fact, eliminating all shrapening and heating labor and equipment. This, however, would require a special chain and would lead us into difficulties beyond our line of endeavor. We were not yet willing to admit that any application of special steels was beyond the realm of possibilities, but for the time being our thoughts were turned to the second of the three suggestions—that of improving our own methods of sharpening and conditioning the bit steel.

In this field we found plenty of possibilities. To begin with, our bit sharpener had been allowed to deteriorate until its ability to function properly was greatly impaired. Upon further investigation we learned that our operation was not alone in this laxity. Before bit conditioning was regarded with importance, it was quite natural that the man in charge of the shop should overlook the bit machine for more pressed needs, as long as it was able to go through the

* Franklin County Coal Company.

motions of sharpening bits. Further, he allowed far more bits to be passed through the process than the machine could possibly sharpen.

Most mine blacksmith shops have coal or oil forges for applying the forging heat to the steel. Constant temperature and similar heating periods on every bit are well-nigh impossible. This, coupled with the fact that bits are plunged into water at varying temperatures after sharpening, tends to produce an assortment of tempers which allows for no two bits in a chain to act alike.

Experiments performed on standard bit steel (and on other steels so near the same analysis that they could hardly be classed as special alloys) making use of a good primary heater, a well-conditioned sharpener, and a controlled heat-treating furnace, have shown that improvements can be made over the ordinary sharpening method. However, the necessary care and skill required to perform such treatment is rarely found in the mine blacksmith shop; nor are many companies willing to spend the considerable investment required to set up the furnaces.

Here again we decide that a good constant temperature primary heater was not out of the question, but that heat-treating furnaces for ordinary bit steel were not to be considered. This decision left for us only the third alternative, that of applying a hard alloy to the point of the bit.

Special Mine Test

Concentrated heat is required to melt and weld the alloy to the bit point. There are two methods of applying the heat to coat the point with the resisting material: Oxy acetylene and electric arc. Because of its lower temperature and simplicity, we tried the former and used it to melt an alloy and tip the bits. There are undoubtedly other materials which might be applied and other methods of applying them, but our experience has been limited to this one operation.

A number of bits were sharpened in the standard manner and a thin coating of the foreign material was applied to the flat top of the bit tip extending back about one-half in. from the actual point.

A certain machine territory in the mine was picked for a comparative test between the standard type of bit and this specially prepared type. The territory chosen for the test was known to enjoy rather uniform cutting conditions from room to room. The power supply was such that the voltage at the face was as uniform as could be obtained.

A certain machine cutting on the day basis was subjected to a careful time study as it went through its daily routine equipped with the standard bits. The following day the specially prepared bits were substituted for the standard type and again the results were carefully noted. The time study comparison follows:

Operation	Standard Bits Time (min.)	Standard Bits Percent	Special Bits Time (min.)	Special Bits Percent	Special Bit Superiority
Setting Bits	78.5	16.65	53.5	11.25	31.80% less
Swing and sumping	58	12.30	46.5	9.80	19.80% less
Cutting face only	226	48.00	285	60.00	20.70% more
Swing and load after cutting	26	5.52	32	6.74	
Machine repair	16	3.41	3	.63	
Moving	57.5	12.22	55	11.58	
Blocked while moving	2	.42			
Machine inspection	7	1.48			
Totals	471	100.00	475	100.00	

Theoretically, the machine crew should work 480 minutes each shift. The sum total as shown in each case is slightly less. This is probably caused by an accumulation of unavoidable errors in keeping the time or by the crews starting late or stopping a little early for some reason. The points illustrated clearly in this time study are that less time is consumed in the unproductive operation of setting bits at the face, less time is required in sumping up and out, and more time is available for actually cutting the face.

On the first day 285 standard bits were used to cut 10 places, representing 1,960 sq. ft. undercut. On the following day only 94 special bits were used to cut 13 places, representing 2,657 sq. ft. undercut.

Each standard bit used undercut 6.9 sq. ft. (approximately 1.8 tons), while each special bit used undercut 28.2 sq. ft. (approximately 7.3 tons), or a ratio of about 4 to 1.

While the time studies were going on, a recording watt-hour meter was keeping track of the power consumption. The results obtained are tabulated below:

Operation	Standard Bit	Special Bit	Special Bit Superiority
Av. sumping time per place	4.60 min.	3.57 min.	22.20% less
Av. cutting time per place	22.60 min.	21.90 min.	3.10% less
Av. K.W. demand	46.00 K.W.	36.05 K.W.	21.60% less
K.W.H. consumed per place	17.18 K.W.H.	14.21 K.W.H.	16.80% less
Total K.W.H. consumed	171.80 K.W.H.	184.85 K.W.H.	20.70% less
K.W.H. per sq. ft. of face undercut	.087	.069	
Av. voltage as read	185 V.	190 V.	

It will be noted that the voltage was fairly constant for the two days, the fact that the average demand was less for the special bits probably accounting for the slight advantage on the second day. The cutting machine has a variable feed speed adjustment and the special bit, offering less cutting resistance, enabled the bar to go through the coal a little faster. It will be noted that there is a distinct saving in the power consumed per square foot of face undercut.

The cuttings, or "bug dust," produced on each day's run were hand screened to determine whether or not the sharper, more durable bits reduced the amount of extreme fines. While there was a slight advantage favoring the special bit, the figures are not regarded with enough importance to enter this discussion.

Having clearly demonstrated that cutting costs might be reduced by the use of the special bit, we next found it necessary to determine what extra costs were encountered in bringing about the improvement. Tabulated results follow:

	Standard Bit 285 used to cut 1960 square feet		Special Bit 94 used to cut 2657 square feet	
	Amount	Per. sq. ft.	Amount	Per sq. ft.
Cost of new bits @ .0222 per each	6.35	.00324	2.09	.00079
Cost of sharpening @ .004 per each	1.14	.00058	.38	.00014
Cost of special application including labor and material @ .0286 per each	....	......	2.69	.00101
		.00382		.00194

The results seemed to indicate that the saving in sharpening labor and the saving in the amount of new steel necessary overcame the extra labor and material necessary to prepare the special bits. We fully realized, however, that our figures were a guess at the best, and that other factors were bound to appear in the process to further increase the preparation cost. Our decision to equip the whole mine with the special bit was made through the belief that as many additional advantages as disadvantages would appear later.

Results of Complete Application

Several months later every machine in the mine was cutting with the new type of bit and the results of the special test were largely substantiated by the figures presented below, covering a period from December 16, 1930, to April 1, 1931. At this juncture it should be stated that the square-foot basis was used on the special test merely because it was easier to record the results. In comparing this test to any other at some future date, the square-foot basis is more equitable because of the variance in seam thicknesses between different parts of a given mine.

However, in checking results over a long period of time, it is well-nigh impossible to use anything but a tonnage basis.

Test covering period during which 203,981 tons of coal were undercut.	
Number of bits sent below	69,972
Number of bits sent out of mine (dull)	64,624
Bits lost or kept in mine (difference)	5,348
Number of bits reground	47,809
Number of bits reheated and prepared again	15,119
Scrapped or discarded on top	4,860
	67,788
Number of new make-up bits used	4,287

It will be noted that each bit sent below accounted for about 3 tons of coal. Using the old type of bit, we used to get only 1 ton per bit sent below. Thus there are now but one-third the number of bits in circulation, and the delivery problems and delays caused by waiting on bits are minimized.

Formerly a bit could be resharpened for use 28 times, while now we find that one bit will take 35 sharpenings before it must be discarded.

Under the old system, every bit had to be reheated and resharpened. Now only 20 to 25 percent need be sent through this process, and the balance are merely touched up on the emery wheel.

The items of cost involved in delivering

(*Continued on page 90*)

CONVEYOR SLOPE OPERATION

at INGLE MINE

By David Ingle, Sr.*

THE mine at which this belt conveyor is operating was originally designed as a slope mine for belt conveyor installation, all mechanical loading, and mechanical preparation at tipple.

The coal is the Indiana No. 5 seam, lying at the point of opening at about 85 ft. below the surface, the seam having an average thickness of 6½ ft., with a hard black slate roof, making for excellent top conditions.

The slope itself was sunk 16 ft. wide, 6 ft. high, and on an 18-degree pitch, and it was sunk at this gradient to a point approximately 30 ft. below the coal, where sufficient space was shot out to permit of the installation of a fairly large hopper for storing the coal from incoming motor trips.

During the first 18 months of development, the coal was hoisted up the slope one car at a time, with a rope, and dumped into a small hopper, whence it was fed to the railroad cars.

The track used in the slope for this purpose remains now on the left side of the slope, and is used for handling supplies and equipment and for hoisting an occasional car of refuse.

The belt conveyor was installed in the right side of the slope, looking down the slope. Between the track and the belt, there was left enough space to install a stairway 20 inches wide, which we use for a walkway in and out of the mine.

The belt is 36 in. wide, of 9-ply construction, with an extra coating of pure rubber on the coal-carrying surface. It is 515 ft. from center of foot pulley to center of head pulley, and is carried on 5-roller carriers on Hyatt bearings, with an occasional guide roller along the sides. It is driven by a 100-hp. motor, the drive being located just under the belt at the ground landing. The belt is equipped with an electrically operated brake to prevent the weight of the coal carrying it backward when the power is shut off.

Coal is brought from back in the mine in 4-ton Sanford Day steel drop-bottom cars. These cars are 14 ft. long, 6 ft. wide and 40 in. high, overall. The coal is dumped automatically into the hopper as the coal train runs over it, without any switching or uncoupling.

The hopper is of 3/16-in. steel plates, and is 34½ ft. long and 16 ft. wide at the top, sloping down to an opening at the bottom 3½ by 5 ft. Under this bottom opening is a steel pan reciprocating feeder, operated by a 5-hp. motor, which feeds the coal from the hopper at any desired rate onto the belt.

The volume of coal fed onto the belt is regulated by the length of stroke on the feeder, and this can be changed from a minimum stroke of 4 in. to a maximum of 12 in. At 4 in. the belt carries 120 tons per hour, at 5 in., about 130 tons per hour, continuously.

The hopper is placed under the tracks, the long way of the hopper making necessary a 39-ft. span under the track. To insure safely carrying the load of train motor and loaded cars, two 24-in. I-beams were installed, grounded at each end into reinforced concrete set into the solid rock. The beams are braced by supports to the side walls, which supports serve for carrying the flooring over the hopper. This hopper has a storage capacity of 85 tons.

The operation of the belt and of the feeder mechanism is controlled by push buttons placed at advantageous points. The belt can be started only from one point, which is at the switchboard located at the ground landing beside the drive motor.

However, it can be stopped instantly by push buttons placed in the tipple, if for any reason the tipple men desire to stop the flow of coal, at the ground landing, or at the bottom of the slope. The feed conveyor can be started from the tipple or from the bottom. The feeder can not, however, be started when the belt is stopped, and any push button which stops the belt also automatically stops the feeder.

One man at the bottom looks after the dumping of the coal, the handling of the supplies on the bottom, takes care of the feeder mechanism, and at odd times oils and looks after the belt rollers, etc. One man is all that is needed between the mine car and the shaker screen. There are no hoisting enginemen, no cagers, no coupling of cars.

The trip rider on the coal train coming into the bottom, jumps off and marks up the number of cars of coal in his trip, on a convenient blackboard, then watches the re-catching of the car doors, as the cars pass over the door-closing device, to see that there is no failure of the doors to latch properly.

Coal is fed from the belt, in the tipple, directly onto the shaker screen, where it is at once divided into plus 3-in. and minus 3-in. The plus 3-in. is allowed to proceed down the screen to where it is further sized and hand picked. The minus 3-in. goes direct into a double Menzies hydro-separator, where the coal is floated out of the impurities and discharged onto a long dewatering screen, which separates it into 3 x 2, 2 x 1¼ and 1¼ by 1/16 sizes. The minus 1/16 coal is lost or rather carried with the water into the sludge tank, where the coal settles to the bottom and is scraped by a flight conveyor up a sloping end, and allowed to fall into a stream of water that carries it away into a settling pond near at hand.

The refuse from the hydro-separator is discharged into a refuse hopper, whence it is hauled away with a motor truck and placed wherever we can find a place to dump it.

Continuous operation is essential to proper preparation of coal for the market. This is true for hand picking, and it is, of course, imperative for proper washing of coal.

The storage hopper with its reserve of coal at the bottom, and the flexibility of the feeder arrangement, enable us to keep the coal moving continuously throughout the day at a rate that keeps the tipple running smoothly and at the same time keeps coal out of the way of the transportation arrangements underground. Very seldom does the tipple run out of coal, and still less frequently are trains delayed on the bottom by reason of delays at the tipple.

Our belt has been in operation only since the last of December. At present we are averaging an output of slightly over 900 tons per day. This coal is hauled to the bottom by one 6-ton locomotive in four-car trips usually, alternating between three loading machine crews. We have 30 mine cars: 4 for the train motor, 4 for each gathering motor, 4 in reserve at each loading machine, and 2 spare cars. We will probably have to buy some more when the haul gets longer, or if we get ambitious and put on another loading crew, which we will do when, as and if, the market clamors for more of our coal.

* President, Ingle Coal Company.

Creosoted ties in section of empty and loaded track, main line in Clyde No. 1 Mine

The Economy of CREOSOTED TIES in Coal Mines

By D. D. Dodge*

DEPENDABLE transportation from the working faces to destination is one of the important factors in the economical production of coal. This means good drainage, satisfactory ballast, adequate ties and switch ties, and rail of the proper weight for the loads to be handled. The track must be well graded and tamped and kept in good condition. The actual attainment of successful and economical operation constitutes a problem which must be solved in the main by the mining industry itself, since many of its problems are peculiar to that industry alone.

Track economy is determined very largely by the service life of ties and their annual cost. This one feature of construction has a most intimate bearing on the ultimate cost of coal production. The experience of W. J. Rainey, Inc., with both treated and untreated ties furnishes data which should be of value to every mine operator. This company operates nine bituminous coal mines in western Pennsylvania, in Fayette, Greene, and Washington Counties, in the Pittsburgh seam of coal. Approximately 4,000,000 tons are produced annually, and a permanent main haulage track of 19.73 miles is operated. This mileage does not include tracks in flats, butts or rooms, which may be considered temporary rather than permanent, as the mine plan of operation has been to go to the boundaries and retreat, room and pillar method of mining.

White oak ties produced locally were used for maintenance and construction during most of the half century of this company's operations. As the supply of this timber began to dwindle, its price increased, and it became necessary to use other oaks and hardwoods. While the results obtained from the white oak ties were generally satisfactory, it became evident that the average life obtained from the mixed hardwood ties was not satisfactory. Their life varied greatly under different conditions of service.

At the Allison mine, in 1926, 1,800 lin. ft. of main haulage track were constructed from the bottom of No. 1 shaft, using extra heavy hewn 6 x 6—6 ft. untreated hardwood ties, principally oak. An inspection in 1931 showed the ties in good condition and probably good for at least three more years, an estimated life of eight years. This piece of track is dry, exceptionally well drained, and in fresh intake air, so that maximum life is obtained from untreated timber. In other locations, where the drainage was poor and the track located at considerable distance from the air intake, decay caused the failure of some ties in 18 months' time.

In 1915 an open-tank dipping plant was installed at the Allison mine, and an experimental track laid at No. 4 right flat on main dip headings. In this track 150 creosote-dipped ties were laid alternately with untreated hardwood ties in the above main haulage track, 4,200 ft. from the bottom of the shaft. In this experiment an average life of four to five years was obtained from the untreated ties, while a substantial increase in life resulted from the dipping treatment.

This open-tank plant was used for several years, but the treatment was hard to control, and the results were not satisfactory, and the plant was finally abandoned. The use of purchased ties treated by standard pressure-vacuum methods in leading treatment plants was begun in

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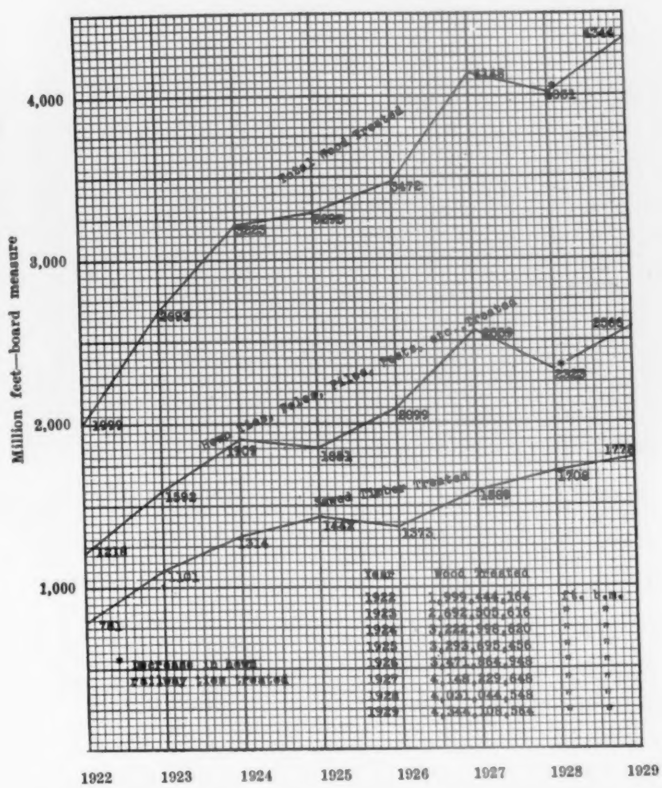
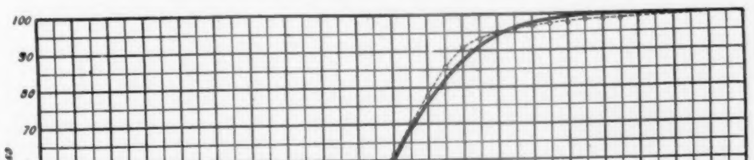
Prepared in cooperation with Mr. Reamy Joyce, vice president, Joyce-Watkins Company.

Standard 250-ft. radius turnout off main line, empty track. All ties creosoted. Clyde No. 1 Mine

Growth of the Wood Preserving Industry 1922–1929





COMPARISON OF ORIGINAL CURVE AND NEW CURVE

Figure 2

1929. Considerable study was first made of wood preservation in general and of the experience of other mines and railroads using treated ties. As a result of these studies, pressure creosoted ties and switch ties are now used in the construction of all main haulage tracks. Since 1929, 28,147 sawed 5 x 6—6 ft. oak ties and 46 sets of 6 x 8 sawed oak switch ties, consisting of 1,598 pieces in all, treated by the Rueping process with a net retention of 6 lbs. of American Wood Preservers' Association 80-20 creosote coal-tar solution per cubic foot of timber, have been used in construction of main haulage tracks. These ties are spaced on 18-in. centers, ballasted with "red dog" and well tamped and leveled. Eighty-pound A. S. C. E. rail is used on outbound load tracks and 60-pound rail on the inbound light tracks. No tie-plates have been used, as the superimposed loads are not considered heavy enough to injure the wood. This feature is being watched closely, and if it is found that mechanical wear is a factor in the service of creosoted oak ties, tie-plates will be installed. Approximately 40 percent of the main haulage tracks are now constructed with creosoted ties and switch ties.

The composite strength values for different species for use as railroad ties were developed by the Forest Products Laboratory and presented before the American Wood Preservers' Association in 1916. The three important factors considered in determining the mechanical suitability of wood for cross-ties are: (1) The bending strength or ability to resist center or end binding; (2) the end hardness, crushing strength, and strength in compression parallel to the grain, which are indicative of the resistance to spike pulling and the lateral thrust of spikes; (3) the side hardness and compression perpendicular to the grain, which indicate the ability to resist rail wear. The weights assigned to each of these values in arriving at the composite figure are shown in *Table I.*

Species of ties adjacent to the properties of W. J. Rainey, Inc., and their composite strength values are shown in *Table II.*

It should be pointed out that the strength values are materially reduced by decay. Untreated material stored for less than two years, and giving evidences of decay only under the most careful inspection, was found to have less than 50 percent of the strength expected. Untreated ties in mines lose their original strength rapidly and are subjected to injury from rail cutting soon after installation.

The most valuable service records of ties available are those of the railroads, the largest users of treated wood in the United States. They were the first to use this material, and the economy they realized from its use led other industries to follow similar practice. Today most large industries are utilizing treated timber for some purpose. The great increase that has developed in the use of treated timber by American industries between the years 1922 and 1929 is shown graphically in *Figure 1.* In the former year slightly less than two billion board feet of treated forest products were used, while seven years later in excess of four billion feet were required.

Records of life of creosoted ties obtained by the railroads give a reliable index of the life and economy which may be expected from the use of treated ties and timbers by other industries, including the mining industry.

The experience of the Big Four Railroad may be cited as an outstanding example of the economies effected by the use of creosoted ties. This railroad, previous to 1904, had used untreated white oak ties for maintenance and construction. At that time a creosoting plant was constructed on the Big Four Railroad and the insertion of creosoted ties started. In 1929, 25 years after beginning their use, 94 percent of all ties in the Big Four tracks were creosoted. The average tie renewals for the five years ending 1909, which was before the railroad obtained the benefit of the treated ties, was 307 ties per mile. The average renewals for the five years ending 1929 were 108 ties per mile—a saving of 199 ties per mile. As the 1929 mileage was 4,147 miles, the total annual average saving for the two five-year periods under comparison was 825,253 ties.

With an average life of 30 years for creosoted ties, it might be supposed that no renewals would occur until the 30-year life was completed. This is not the case, however. The first renewals in any lot of ties will occur at approximately one-third their average life, and the last renewals will occur at approximately one and two-thirds times their average life. These data are interesting and important

in anticipating the results to be obtained from creosoted ties in track.

Figures 1 and 2 are curves developed by Dr. J. D. MacLean, of the U. S. Forest Products Laboratory. *Figure 2* shows the relation between the ties replaced and their average life. *Figure 3* provides a means of determining the probable life of ties from the percentage of ties replaced.

The comparative economy of treated and untreated ties may be calculated by the A. R. E. A. formula below when the initial costs and service life of the ties are known or assumed.

$$A=Pr\frac{(1+r)^n}{(1+r)^n-1}, \text{ where}$$

A=Annual charge,
P=Amount of initial investment,
n=Number of years in the recurring period (the average life of the timber), and
r=Rate of interest expressed decimally.

Computed from this formula, the annual charge due to an initial investment of $1 for periods varying from 1 to 60 years is as follows:

Interest rate	Term of years 1	2	3	4	5	10
6 percent ...	1.060	.545	.374	.289	.237	.136

Interest rate	Term of years 15	20	25	30	40	50	60
6 percent	.103	.087	.078	.073	.066	.063	.062

Assuming the following initial costs:

	Untreated	Creosoted
F. o. b. mine...............	$0.29	$0.70
Handling and insertion......	0.67*	0.67*
Tie in place..........	$0.96	$1.37

* Cost of installation is experience in new construction. It is recognized that the cost of spot renewals in existing track is considerably greater.

Then the annual cost for treated and untreated ties for the periods of life assumed will be as follows:

Untreated ties:	
4-year life	$0.277
5-year life	0.228
Creosoted ties:	
20-year life	0.119
30-year life	0.10

If we assume a life of five years for the untreated ties, with the annual charge of $0.228 per tie, and a life of 20 years for the creosoted ties, with the annual charge of $0.119 per tie, there is a saving in favor of the creosoted ties of $0.109 per tie per year, or a saving of $383.68 per mile of track per year.

If, on the other hand, we assume a four-year life for the untreated ties, with the annual cost of $0.277 per tie, and a 30-year life for the creosoted ties, with the annual cost of $0.10 per tie, there is a saving in favor of the creosoted ties of $0.177 per tie per year, or a saving of $623.04 per mile of track per year.

(*Continued on page* 86)

TABLE II—COMPOSITE STRENGTH VALUE FOR TIES

Hickory	1,354
Sugar maple	1,128
Sweet birch	1,079
Red oak	1,001
Beech	973
Black cherry	840

Standard crossover from loaded to empty track. Crossovers are located at each turnout and for the first 2,500 ft. from mine portal at intervals of 500 ft. Creosoted switch and cross ties, Clyde No. 1 Mine

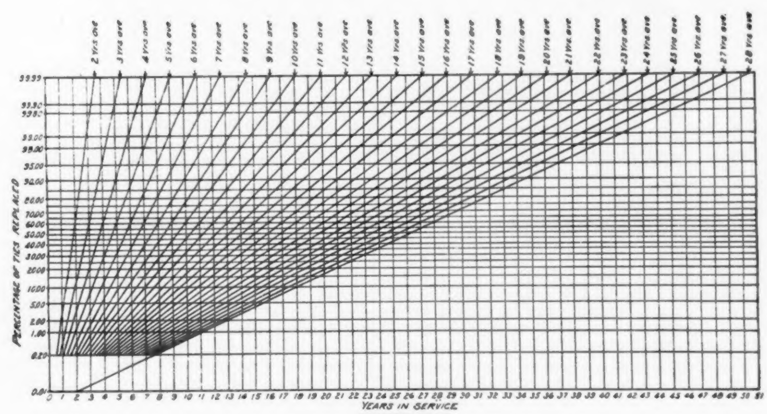


CHART SHOWING AVERAGE LIFE TO BE EXPECTED AFTER VARIOUS PERCENTAGES OF RENEWALS HAVE BEEN MADE

Figure 3

TABLE I—BASIS FOR DEVELOPMENT OF A COMPOSITE STRENGTH FIGURE FOR TIES

Mechanical property	Relative weight used in forming composite figures	Relation of mechanical property to use of species for crossties
Static Bending:	Percent	These properties involve the strength when used as a beam. They are of primary value in determining the resistance offered to breaking caused by "center binding."
Modulus of rupture...............................	14.3	
Fiber stress at elastic limit.......................	7.1	
Impact Bending:		
Fiber stress at elastic limit.......................	7.1	
Total	28.5	
Compression Parallel to Grain:		These properties involve the resistance offered to a compressive force exerted lengthwise along the grain and are of value in indicating the resistance offered to lateral pressure on spikes.
Fiber stress at elastic limit.......................	7.2	
Maximum crushing strength.......................	14.3	
End hardness	10.0	
Total	31.5	
Side hardness	20.0	Indicates the resistance to rail wear, abrasion, etc.
Compression Perpendicular to Grain:		
Fiber stress at elastic limit.......................	20.0	
Total	40.0	
Composite figure	100.0	

AIR-SAND PLANT of the Allegheny River Mining Company

By R. M. Shepherd*

THE officials of the Allegheny River Mining Company, Kittanning, Pa., after inspecting the operation of a small model Air-sand coal cleaner in the Carnegie Institute of Technology, Pittsburgh, in the spring of 1930, were impressed with the underlying idea of cleaning coal with dry sand and air, and after due deliberation decided to give it a test on an operating basis. To this end arrangements were made with the inventor, Thomas Fraser, representing the Hydrotator Company, of Hazleton, Pa., for the installation of one 4-ft. separator with necessary screens and accessories in the tipple at the Cadogan mine. This installation proved so promising that an additional 6-ft. separator was lately placed in operation.

The basic principle of the process is the difference of gravity between coal and slate and other impurities, and is in reality a sink-and-float process.

Dry sand passes through a 12-mesh screen, where it is subjected to fine jets of air, blown through a porous stone slab in the bottom of the separator, having an upward movement through a bed of sand. This has the effect of causing the sand to float the light material on top, while the heavy materials sink to the bottom. The coal flows over a roller onto a de-sanding screen, where it is separated from the sand and delivered to the car or bin, as the case may be. The slate and other heavy materials sink to the bottom of the separator and pass out, together with some coal that remains with the refuse, and all of the sand. It is then delivered to an elevator, where it is elevated and returned to another compartment of the separator, for recleaning, or recovery of the coal that sank with the refuse in the first cleaning. The coal so recovered passes over and into the first separator compartment, and then passes out with the clean coal to the car or bin. The refuse matter from this second operation passes onto a de-sanding screen, where the sand is recovered, and the refuse is transported to the refuse bin or dump. The sand is used over and over again. The air is delivered by a motor-operated blower through pipes to the porous stone slab. These pipes have air gates to regulate the flow of air, and consequently the pressure under the porous stone slab, which effects an efficient regulation of the fluidity of the sand bed. The separator is stationary and the de-sanding screens are shakers. In the present installation only nut and pea sizes of coal are being cleaned. The nut passes through 2-in. and over 1-in. lip screen; the pea passes through 1-in. lip screens and over ⅝-in. vibrator screens. The arrangements are so that either size of coal can be cleaned in either separator, or mixed and cleaned together on the large separator, while the recleaning is accomplished on the smaller separator.

When the raw coal is dry, or only partly moist, the separation of the sand from the coal on the de-sanding screen is nearly perfect; but when the raw coal is wet there is a tendency for the sand to adhere to the coal, also for the sand to cake and pass over with the coal. To overcome this it is necessary to heat the sand to dry the film of moisture adhering to the coal enough to shake the sand loose on the de-sanding screens, and prevent the caking of the sand. A temporary coal-burning furnace was constructed for this purpose, through which the sand and refuse passed, and although this method of drying was efficient, it was cumbersome and unsafe on account of the fine coal dust present; therefore a heating system of steam pipes, heating air and sand, was devised to eliminate the furnace.

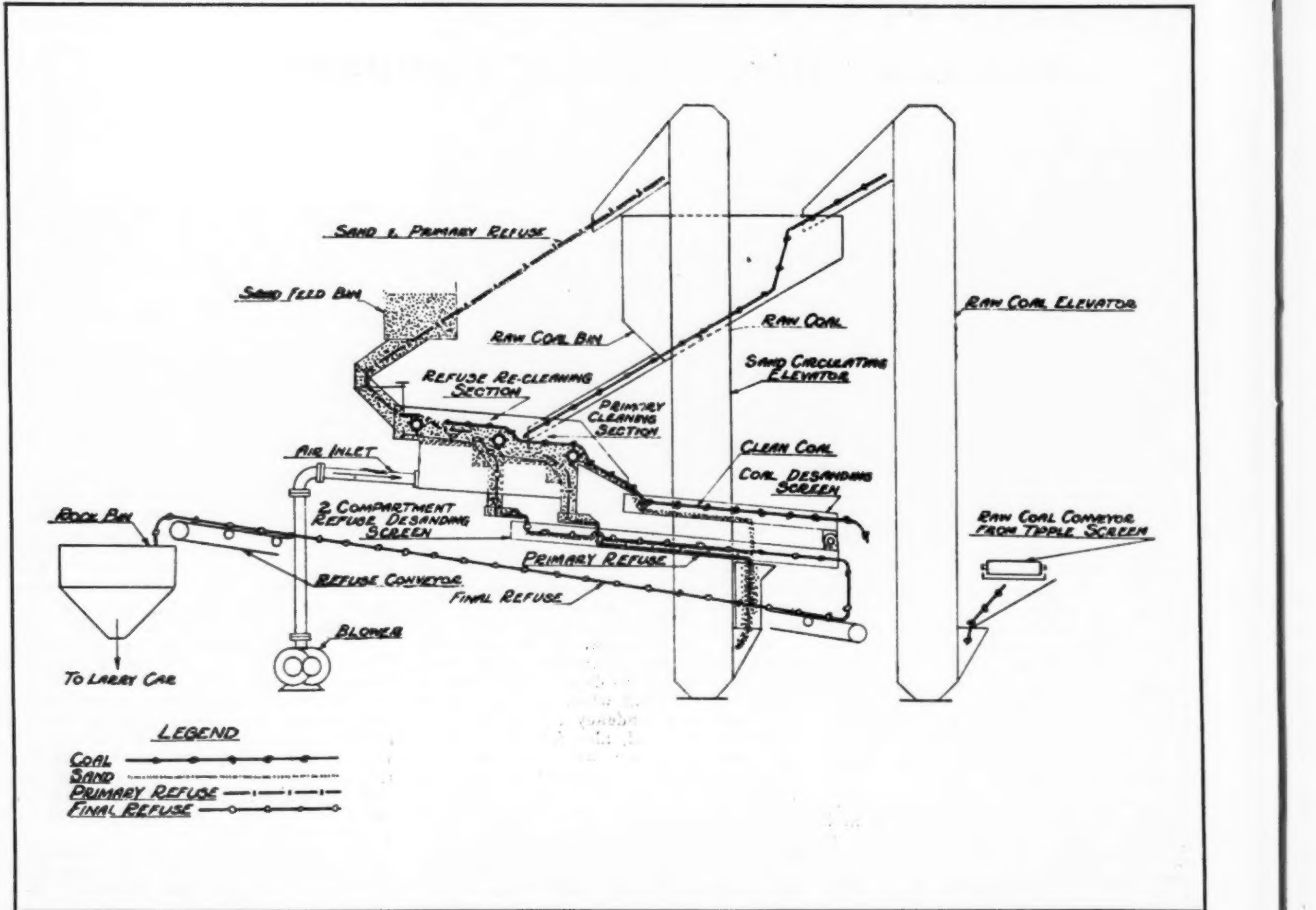
The raw coal is elevated into a bin, where it gravitates into the separator. There it mixes with the sand, kept in a fluid state by the air blast. The sand is also elevated to a bin above the separator, and gravitates into the rear, pushing the mass forward. The light material, or coal, passes over a roller onto the de-sanding screen, while the heavier materials sink to the bottom with the sand; they are then carried to the sand elevator, where they pass into the second compartment of the cleaner, as previously described.

The capacity of the 4-ft. separator is approximatelely 30 tons an hour, and the 6-ft. separator is 50 tons. When working under normal capacity it requires approximately 3 pounds of make-up sand per ton of clean coal. The sand in the circuit is about 5 tons for the two separators, and is circulated at a rate of 2½ tons per ton of coal cleaned. The sand in the separator must be kept dry at all times and screened through a 12-mesh screen. The air is filtered before it enters the porous stone slab to prevent clogging of the fine pores. The air is delivered under the slab at a pressure of 12-in. water gage, more or less, at the rate of 1,500 cubic feet per minute, for both separators, and is furnished by a 10 by 35-in. Root type blower, driven by a 15-horsepower motor.

The apparatus consists of separator, de-sanding screen, blower, air filter and air pipes, coal and sand elevators, degradation screens, refuse conveyor, various chutes, sand drying apparatus, motors, and driers, and can be installed complete, with building, for approximately \$50,000 per 100 tons of clean coal per hour. The operation of the plant is simple. Only two men are required to care

The Air-sand separator without de-sanding screen

* President, Allegheny River Manufacturing Company.



for the process, including the generation of steam for the drying of the sand, heating the air, handling sand, and keeping the place tidy. These two men should be able to handle almost any size of plant, and consequently vary the cost according to the output. From our observation and records, the cost of operation per 100 tons of clean coal per hour is approximately as follows:

2 men at 50c per hour	$1.00
Power, 30 kw. at 1½c per hour	0.45
Interest	1.80
Depreciation	2.36
Repairs and renewals	0.75
Steam generation	0.05
Oil and grease, make-up sand, etc.	0.01
Total	$6.42

or 6.42c per ton of cleaned coal. Because nearly 100 percent of the coal mined at this mine is mechanically loaded, the smaller sizes of the product carry a large amount of slate, pyrite and other impurities, as shown in the following sink and float tests:

Product	Float 1.45 sp. gr., %	Float 1.45 to 1.60 sp. gr., %	Sink 1.60 sp. gr., %
Raw nut	83.0	3.0	14.0
Cleaned nut	97.6	1.8	0.6
Raw pea	85.1	3.3	11.6
Cleaned pea	95.9	2.8	1.3
Primary refuse nut	45.4	11.3	43.3
Primary refuse pea	43.4	16.8	39.8
Final refuse nut	2.8	6.2	91.0
Final refuse pea	3.2	4.7	91.5

Proximate analyses of the raw and cleaned coals give the following results:

Proximate Analyses of Raw and Cleaned Coals Percent

Product	Moisture	Valuable Matter	Fixed Carload	Ash	Sulphur	B. t. u.
Raw nut	0.70	35.18	48.98	14.14	4.31	12,887
Cleaned nut	0.96	37.46	56.10	5.48	2.08	13,999
Raw pea	1.04	33.87	50.10	14.99	3.94	12,577
Cleaned pea	1.16	34.71	55.12	9.01	2.96	13,600

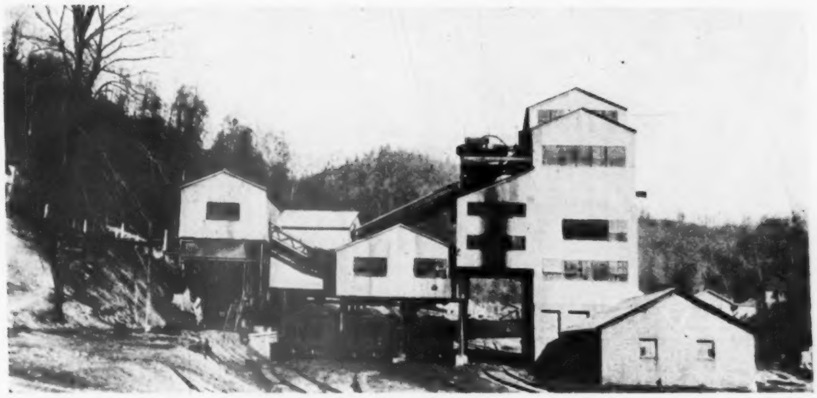
The process is still in the experimental stage, and while we feel much encouraged with the results obtained, we also feel that there are further possibilities which can be worked out as we go along.

Safety With Electrical Equipment

(Continued from page 60)

of the equipment. It is not enough to tell the man of the various hazards when he enters the employ of the company, but safe methods and safety should be kept before him at all times. This may be done by means of safety meetings which are held in each section once a week, or as may seem best to those in charge, safety meetings on the surface to be conducted by the safety engineer, members of your own Electrical Department, or representatives of the manufacturers.

If possible, some incentive should be given to those who are responsible for the elimination of extraordinary unsafe methods or equipment. We believe that accidents from electrical causes will be greatly reduced, if not entirely eliminated, when each man makes it his personal problem to see that he operates and maintains his equipment not only having in mind his own safety, but the safety of his fellow workmen, and the investment of the company for whom he is working, and we believe that education is the only method of accomplishing this.



The Nellis washer

WASHING PRACTICE at Nellis Mines

By E. H. Shriver*

THE principal coal mines of the American Rolling Mill Company are at Nellis, Boone County, W. Va. The coal mined is the No. 2 gas seam. It averages 48 in. in height and contains a bone parting, 2½ to 4 in. thick about 10 in. from the top. There is also a streak of high ash splint coal near the bottom. The splint coal averages from 3 to 4 in. in thickness except in the "swags" where it is somewhat thicker and the ash much higher. The roof is a very tender slate which occasions more or less difficulty in mining. The bottom is fireclay.

The mines at Nellis were developed because of the gas-making properties of the coal, the low sulphur content, and the extremely high fusing point of the ash. As the mines increased in size with the consequent increase in the percentage of pillar coal, adequate preparation of the coal by the ordinary hand picking methods became almost impossible. As may be seen from the following analysis of the coal as mined, the ash in the smaller sizes is fairly high.

Analysis of Three Sizes of Coal, Percent

	4½-in. lump	1 x 4½-in. egg	0 x 1½-in. nut and slack
Volatile matter...	38.00	35.00	33.00
Fixed carbon.....	56.00	54.50	50.00
Ash	4.00	7.50	13.00
Moisture	2.00	3.00	4.00
Total	100.00	100.00	100.00
Sulphur	0.73	0.90	1.20

In 1926 the Coal Department of American Rolling Mill Company began to investigate the possibility of obtaining better preparation through some system of mechanical cleaning. Complete washability studies were made on the 0 to 4 in. coal. From the sink and float data thus obtained, it was found that Nellis coal was amenable to washing and that good results could be expected in both ash and sulphur reduction by mechanically cleaning the coal.

With the above information at hand, thorough examinations of all the different types of plants, then in commercial operations, were made for their possibilities in: (1) ash reduction, (2) sulphur reduction, (3) recovery efficiency, (4) moisture in nut and slack coal, (5) flexibility of operation, (6) cost of operation, (7) first cost.

During the year 1928 it seemed that some pneumatic cleaning system would be the only possible solution to the problem as the Combustion Department of American Rolling Mill Company had insisted on dry nut and slack for boiler fuel. The Carpenter dryer was being developed, but had not yet proven particularly successful in drying fine coal to the point demanded by coal consumers. However, the short comings of the dryer were largely corrected and with the "bugaboo" of wet slack coal out of the way, the company decided to purchase a Rheolaveur plant because it seemed to fill the requirement more closely, perhaps, than did the other commercial methods in use at that time.

The contract for a 200 ton per hour cleaning plant capable of cleaning the minus 4-in. coal from Nellis mines was let to the Koppers Rheolaveur Company in May, 1929. At the present time all minus 4½-in. coal is washed.

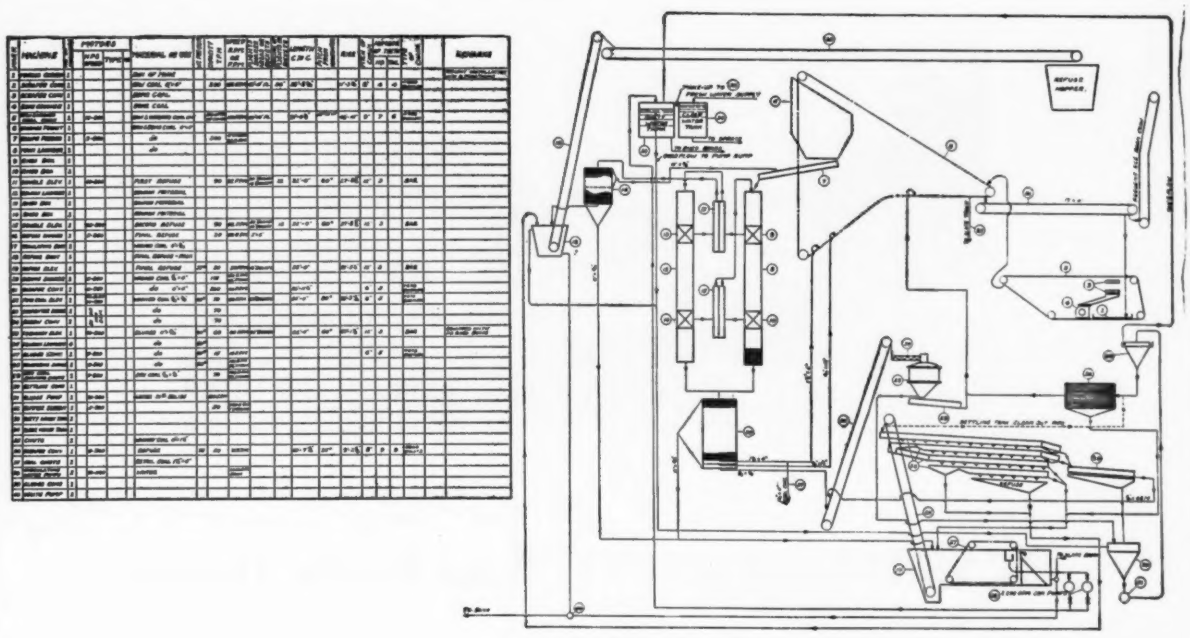
Construction started in August, 1929, and the plant was completed in February, 1930.

The design of the plant was different from any Koppers-Rheolaveur washer containing a free discharge or fine coal system, previously built for cleaning American bituminous coals, in that no Dorr thickeners or filters had been included. Because of this change in design some difficulties were encountered during the first few months' operation of the plant. These difficulties were finally largely overcome and the plant was turned over to the American Rolling Mill Company on July 1, 1930, for independent operation.

The building that houses the cleaning plant is extremely well constructed of structural steel with Armco corrugated siding and roofing and concrete floors throughout. An abundance of metal sash windows provide good light in practically all parts of the building. The building rests on a massive concrete foundation, designed to contain the main settling tank, the final refuse hopper, the pump room and the roughing laboratory. The equipment is constructed of Armco blue annealed sheets, where possible. High starting torque Westinghouse squirrel cage motor for full voltage starting was furnished. An open type switchboard providing complete control and protection for the motors placed in a central control room was furnished. All starters were full voltage except starter for Carpenter dryer where reduced voltage starter was furnished. The proper sequency interlocking of motors was provided on the board. The circulating water and make up water pumps were furnished by Allis Chalmers and the sludge pumps by the Morris Pump Works.

As may be seen from the accompanying flow sheet, the coal from the mine is dumped onto the Marcus screen tipple as was previously done, the minus 4½-in. coal passing through the screen plates onto a cross raw coal scraper conveyor which drops it onto the upper strand of the raw and washed coal conveyor. The coal is carried to the top floor of the washer building and is dumped into a 65-ton storage hopper. The coal from the hopper is fed into the main washing launder by means of a plate feeder. The main washing launder contains two Rheo boxes, the upper one having a countercurrent flow of water. The refuse, containing some high ash coal, goes through

* Koppers Coal Company.



Flow Sheet
Rheolaveur Washery

the first Rheo box of the main launder, is elevated in a double compartment elevator and discharged into the main re-wash launder. Any refuse passing over the first Rheo box is caught in the second box and is elevated in another double compartment elevator, is discharged onto the raw coal plate feeder and is again washed in the main launder.

The re-wash launder also contains two Rheo boxes, the upper box having a counter flow of water. This upper box is known as the slate box as all the refuse from the sealed discharge plant is eliminated at this point. The refuse is elevated to a refuse shaking screen containing ⅝-in. and 5/16-in. round hole screens. The material passing the ⅝-in. screen is discharged onto the plate feeder and is sent back to the main launder for its third washing. The material passing the 5/16-in. screen is dropped through a pipe to the settling tank and from there goes to the free discharge plant to be re-washed. Any refuse which goes over the slate box is caught in the lower box of the re-wash launder and is elevated through the double compartment elevator from which it is discharged into the main launder for another washing.

The clean coal from the main and re-wash plant is carried through a launder to the main sizing shaker screens. The 1½ in. x 4½ in. egg coal is screened on the upper deck and goes directly to one side of the lower strand of the raw and washed coal conveyor, is carried back to the tipple and loaded over a boom onto the cars. The ¾ in. x 1½ in. nut coal is screened on second deck and goes to slack side of raw and washed coal conveyor. The 5/16 in. x ¾ in. coal is screened out on the third deck and is discharged into the boot of the fine coal elevator, which feeds it to the Carpenter dryer. The 0 x 5/16 in. material passing the third deck of the sizing screen, goes to the settling tank from which it is collected by a drag conveyor and fed into the boot of the free discharge elevator which discharges it into the fine coal or free discharge plant.

The free discharge plant contains two clean coal launders, a middling launder and a slate launder. Each of these launders contains several Rheo boxes, some of which are equipped with counter current water. The overflow from the two top or A and B launders goes directly to a fine coal dewatering screen which is equipped with 48-mesh wedge wire screens. The 48-mesh by 5/16-in. coal is dewatered to about 12-14 percent moisture and then passes to the boot of the fine coal elevator and from there to the Carpenter dryer, where it is dried with the 5/16 in. x ¾ in. coal from the sizing screens. The overflow from the middling or C launder and from the slate or D launder passes back into the settling tank for recirculation. The final refuse passing through the slate launder goes directly to the refuse hopper.

The underflow from the 48-mesh wedge wire screen is passed to a cone, the overflow going to the settling tank and the underflow going to a sludge pump which pumps the thickened material to another cone at the top of the washer. The effluent from the Carpenter dryer is also passed to the first cone. The upper cone dewaters the fine coal to about 50 percent moisture, the overflow going to the constant head tank while the partially dewatered underflow is fed to a two-deck Zimmer screen which is equipped with 80-mesh monel metal cloth. The Zimmer screen dewaters this fine material, amounting to some 8 tons per hour, to about 20-23 percent moisture. The material going over the screen is dropped directly onto the slack side of the raw and washed coal conveyor where it is mixed with the ¾ in. x 1½ in. nut coal and the dried coal from the Carpenter dryer and carried to the car.

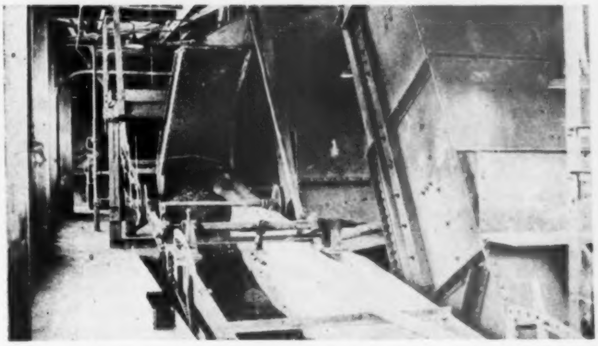
The underflow from the Zimmer screen is sent back to the settling tank at this time. However, it is probable that this material will eventually be sent directly to the refuse hopper as it is high in ash and sulphur.

The final refuse is elevated from the refuse hopper to a drag conveyor which carries it to a slate hopper in the tipple from which it is hauled to the slate bank.

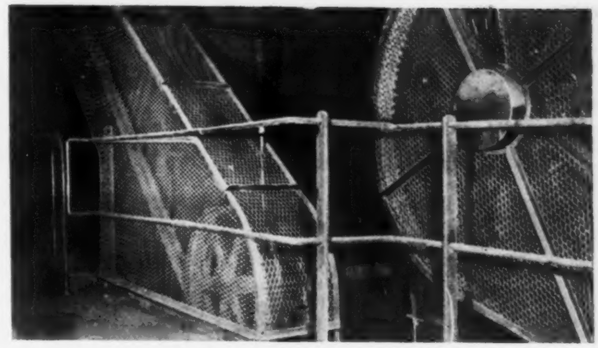
One of the characteristics of the Rheolaveur process of cleaning coal is the necessity for maintaining a slate bed in each launder at all times. In the sealed discharge plant this bed may be as much as 6 or 8 in. deep; in the free discharge plant the bed is generally about 2 or 3 in. deep. These beds are maintained by baffle boards or barrages placed crosswise in the bottom of the launders. When the Nellis plant was designed it was thought necessary to send all of the pickings, both slate and bone, from the lump picking table to a roll crusher and from there to the cleaning plant in order to provide sufficient bed material. After operation of the plant had started it was found the regular feed to the plant contained sufficient bed material. It is probable the pickings from the lump boom will be eventually sent directly to the slate hopper.

Control of the sludge or minus 80-mesh coal has been one of the greatest problems in connection with the wet washing of coal. It occupies the same position as does the control of the dust in the dry cleaning plant. With the present scheme of operation at the Nellis plant, no trouble of any consequence has been experienced since the plant was taken over for operation.

As about 8,000 gallons of make-up water is used in the Nellis plant per day, there is, consequently, about the same



Main washing launder



Elevator guards

amount to be disposed of each day less the moisture contained in the final refuse, which is about 15 percent of the refuse. The waste water is taken largely from the final refuse hopper and is pumped to the head of a slate bank and discharged. The slate bank acts as a natural filter and prevents the contamintion of the streams. The water discharged each day contains from 4½ to 6 tons of material analyzing from 15 to 25 percent ash and a large percentage of sulphur. Any overflow water from the settling tank is also pumped to the bank and is included in the 8,000 gallons mentioned above. The loss of practically all of the settling tank overflow water could be avoided by more careful control of the make-up water input and possibly by the synchronization of the make-up water pump and the sludge pump discharging to the bank.

The ease of operation of the Nellis plant has proven to be one of its best features. The operating personnel consists of: 1 washer and tipple foreman, 1 operator on sealed discharged plant, 1 operator on free discharge plant, 1 greaser and clean-up man.

The control personnel consists of: 1 chemist, 1 coal sampler.

The foreman is also the mechanic and does the routine repair work assisted by the greaser. At times the mine mechanical repair force has been called on for special jobs. The operators are two young men who had been slate pickers when hand preparation was used.

The results obtained in the operation of the Nellis plant has, in the writer's judgment, amply proven the claims of the Koppers Rheolaveur Company as to efficiency, regularity of product, moisture elimination, cost of operation, etc.

When the washing operations started it was decided to set the plant to wash the egg coal to 5.5 percent ash and the nut and slack coal to 7.0 percent ash, these percentages apparently showing the most economical results according to the washability curves. This is the set up now in use. If desired the egg coal could be washed to 5 per cent ash and the nut and slack coal to 6 percent ash.

Free discharge launders

One of the features stressed by the Combustion Department was the necessity of regularity in the percentage of ash as well as reduction in ash. Due to the band of high ash splint coal, mentioned above, being largely cut out by the machines practically all of this material is dumped, in the morning. The nut and slack coal washed at this time contains a larger percentage of coal approximating the washing gravity of 1.55 than is the case during the rest of the day. As a result the ash content of the nut and slack coal washed in the morning is a little higher than is the case of the coal washed in the afternoon. In other words the washability curve of the morning coal differs a little from the curve of the afternoon coal. In spite of this condition the regularity of the ash in the washed coal has, in general, been very satisfactory to the Combustion Department of the American Rolling Mill Company.

The average analysis of all nut and slack coal shipped to the Middletown, Ohio, plant during September and October, 1930, were 7.32 percent ash, 5.43 percent moisture, and 0.86 percent sulphur.

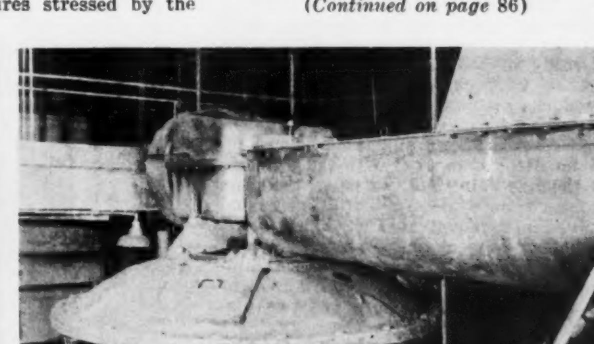
The ash during the above two months was 0.32 percent higher than the agreed figure of 7 percent. The sulphur content was reduced 0.34 percent and the moisture was lower than that of raw nut and slack shipped from many mines. The ash and sulphur content of the egg coal has proved so satisfactory, it is my understanding that regular analysis of this grade of coal has been practically discontinued at Middletown.

The results obtained in actual washing practice has closely approached the theoretical results, as shown by the washability curves.

Following are the results obtained in

(Continued on page 86)

Zimmer screen



Carpenter drier

The New Washing Plant of the Big Vein Coal Company

By Chas. Gottschalk*

A MODERN coal washing plant is now nearing completion at the Big Vein Coal Company's mine near Buckskin, Ind. This location is about 22 miles north of Evansville. A contract was awarded the Roberts & Schaefer Company of Chicago for the design and construction of the washery, and alterations required in the old tipple. The new arrangement is one that has been designed to meet the particular preparation requirements of a mine which loads all coal with loading machines.

The extent of the economic advantage of a positive cleaning plant to treat the smaller sizes of coal depends upon the characteristics of the coal seam, nature of the roof and floor, method of mining, and the markets served.

The selection of the kind of a plant to be installed at Buckskin followed two years of study and investigation given to the problems, after which it was decided by the management of the company that in order for the plant to function economically over an extended period of time, it must conform to the following conditions: *First*, the washing plant must be designed to receive either wet or dry raw coal. *Second*, the plant must be designed so as to perform efficient washing of the finer as well as of the coarser grades of screenings without entailing undue loss of coal in the refuse. *Third*, the minus quarter inch coal must be dewatered to a greater degree than possible over dewatering screens. *Fourth*, the cost of the plant must be consistent with benefits to be derived.

In order to select adequate equipment it was first necessary to determine and consider the nature of the cleaning problem that this mine presented. This meant a study of the coal seam in place, together with changes occurring in the product as a result of mining and transportation from coal-face to tipple. The coal seam is known on the market as Indiana Fifth Vein Coal. On the property of the Big Vein Coal Company, the seam ranges from 6 to 8 ft. in thickness, and is under excellent roof of black slate. The floor is a shale-like fireclay which is easily disintegrated by mechanical mining operations to the especial detriment of the screenings. There are two distinct bedding planes in the seam. At times these planes are only indicated by a trace of mother coal. More frequently, however, two high ash partings occur. One of these seldom becomes more than one-half inch thick, but the other may increase to a thickness of 2 in. These partings vary in color from black to dirty gray, and are noticeably heavier than coal. The other principal impurity to contend with is pyrite which occurs in various sizes and shapes.

Although this seam affords the main source of coal production for the State of Indiana, and has been marketed by most of the mining companies without recourse to washing, it is generally acknowledged that the demand for better preparation must be met.

Experience in mechanical mining and loading at Buckskin developed the fact that various mechanical operations increase the breakage of impurities causing the degradation therefrom to concentrate in the fines. The drier the coal, the more noticeable the concentration of the heavier particles. The advantage of washing, however, is more than an offset to this objection to mechanical loading, because of the premium placed on all sizes and combinations of the washed product.

After a study of the screen analysis of the coal, the washer capacity was fixed at 125 tons per hour of 2 in. slack coal. The primary treatment unit in the washer is a twin tandem hydro separator which receives the 2 in. x 0 feed and from which two refuse products are discharged. The primary refuse is to be a final product and is to be wasted, while the secondary refuse goes to another hydro separator cell and is rewashed, making more final refuse and salvaging any float coal remaining from the primary washing.

A separate secondary plant is provided for the treatment of the fine coal. For purposes of designing, the fine coal has been considered as ¼ in. x 0, but it is expected that the feed to the fine coal washer will probably be even smaller than this. The discharge from the hydro separator plant passes over two sets of screens and from the second of these, the water and fine coal go to the secondary plant, and after the removal of the excess water, is fed on to four Deister-Overstrom tables. Dewatering of the clean coal from the tables is effected by a combination of shaker screen and centrifugal dryer, the prepared fine coal be-

Phantom view of primary and secondary cells of a twin hydro separator

* Vice President, Big Vein Coal Company.

ing finally delivered to the same loading point as the slack from the hydro separators.

The following description of the progress of material through the plant will assist in visualizing the complete process.

The coal is dumped from the skip into a raw coal hopper from which it is removed at a constant rate by a reciprocating feeder. From the feeder the coal is fed onto a shaker screen, where it is sized into three products, 4 in. lump, 4 in. x 2 in. egg, and 2 in. x 0 slack.

The lump coal is discharged onto a lump picking table and loading boom, where it is handpicked. Similarly the egg coal is handled on an egg picking table and boom. These booms either load into cars or discharge onto a tipple mixing conveyor. In the latter case, the lump can be loaded on the egg track with the egg coal, or can be discharged to a lump coal conveyor which delivers the coal to a bone coal conveyor and finally to a single roll crusher. Here the lump is crushed to 2 in. and under, then fed to crushed lump conveyor leading to the top strand of a two-compartment mixing conveyor which conveys the crushed lump to a loading chute on the slack track. Thus the lump can be loaded on the lump track, egg track, or when crushed, on the slack track.

The pickings from the lump and egg can be graded, when required, as refuse and bone coal. The refuse is thrown into a refuse pickings conveyor for discharge into refuse bin, thence to be hauled away by mechanical means to the refuse dump. The bone coal is thrown onto the bone coal belt conveyor and discharged to the crusher. When this is being done, the crushed bone is fed to an elevator which elevates and discharges the crushings into the raw coal hopper for recirculation through the plant.

This provision makes it possible to salvage a large amount of coal adhering to the bone or slate which would otherwise have to be thrown away to protect the quality of the lump coal. It also provides a means by which lump coal or the lump and egg can be reduced to the smaller sizes in case of a falling off of the demand for this large coal.

The 2 in. x 0 coal runs by chute from the shaker screen to the top strand of a raw slack scraper conveyor which conveys and elevates the coal to a chute where the coal is split into two portions and fed to a battery of hydro separators.

The hydro separator battery consists of five cells, divided into two primaries, two secondaries, and a refuse rewash cell.

Each half of the 2 in. x 0 coal is fed into a primary cell. Here it is washed and two products made, clean coal and refuse. The refuse is carried by the hydro refuse conveyor to the bottom strand of the raw slack scraper conveyor, and by this latter unit is conveyed to the refuse pickings conveyor, for disposal in the tipple refuse bin. The clean coal from the primary cells passes into the secondary cells and is rewashed, clean coal and reject products being taken from each cell. In order to ensure complete cleaning of the coal, the secondary reject is cut well into the washed coal fraction. This reject from both secondary cells is then rewashed in the rewash cell, where any coal is recovered. The same disposal is made of refuse from the fifth or rewash cell, as in the case of the refuse from the primary cells. The recovered coal from the rewash cell, together with the washed coal from the secondary cells, flows by sluice to a coarse coal dewatering screen of the shaker type. This screen also sizes the coal into 2 in. x 1¼ in., 1¼ in. x ¾ in. and ¾ in. x 0.

The 2 in. x 1¼ in. coal can be delivered by suitable chutes to either the lump coal boom or to the bottom strand of the two-compartment mixing conveyor for loading on either the egg track or the slack track.

The 1¼ in. x ¾ in. coal is fed by the screen to the bottom strand of the two-compartment mixing conveyor which unloads it as desired to either a 1¼ in. x ¾ in. belt conveyor which carries the coal to a loading point on the egg track, far enough up track to prevent interference in loading two sizes on one track, or to the egg boom, or to the slack track loading chute.

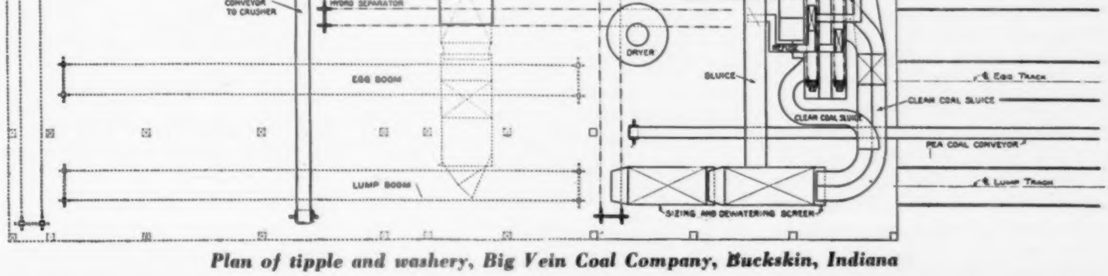
The ¾ in. x 0 coal together with the water from the hydro separators is sliced to an Arms horizontal vibrating screen located over the slack track. The water and ¾ in. x 0 coal is screened at ¼ in. on the Arms screen. The dewatered ¾ in. x ¼ in. is discharged into a chute so arranged that the coal can go to either the slack coal loading chute, or to the top strand of the two-compartment mixing conveyor. In the latter case it can be discharged onto the egg loading boom.

From the Arms screen the ¼ in. x 0 coal and water flow to a battery of classifiers built as inverted pyramids. The battery consists of two units, each having one primary and one secondary tank. Here the coal is classified and settled. The overflow water from the secondary tanks with such extremely fine wash material as may be carried and which incidentally is of high ash content, is sluiced to an adjacent pond where the water gradually clarifies and becomes available for make-up water for the plant.

The underflow from the classifier tanks is used as a feed for four Diester-Overstrom concentrating tables, each tank feeding one table.

From the wet tables, two products are taken—clean coal and refuse. Both products are sluiced to a Parrish type fine coal draining screen, fitted with a longitudinal partition to keep the clean coal and refuse separated. Draining is effected by small openings in wedge wire cloth, and this coal is discharged from the screen to the boot of an elevator which elevates the coal to a height suitable for delivery to a chute leading to a Carpenter centrifugal dryer.

The coal is further dewatered in the dryer, and as the dryer is located over the slack track, the coal passes directly from the dryer to the slack car together with the ¾ in. x ¼ in. coal from the Arms screen, and such other sizes of coal conveyed to the slack chute, by the two-compartment mixing conveyor. Or, the ¼ in. x 0 coal can be run from the dryer to the top strand of the two compartment mixing conveyor for loading out on the egg boom.



Plan of tipple and washery, Big Vein Coal Company, Buckskin, Indiana

The table refuse is fed by the dewatering screen to the boot of a refuse elevator which elevates and discharges this material to a chute leading to the bottom strand of the raw slack conveyor from where it goes to the pickings conveyor and refuse bin.

The bulk of the water in the ¼ in. x 0 coal from the tables passes through the fine coal draining screen into a sludge tank. The overflow water from the sludge tank flows to a pump supply tank and is pumped to the tables and hydro separators. The solids which settle in the sludge tank are taken to one end of the tank by a scraper conveyor and the sludge removed by a sludge pump, which pumps this material to the sluice from the classifiers to the settling pond. Part of this water is put into the plant in the form of sprays on the various screens.

The possible loadings from the plant are numerous and the entire arrangement has been so designed as to give the utmost degree of flexibility in control of percentages of fines entering the various marketable products.

The Economy of Creosoted Ties

(*Continued from page* 78)

It is expected that the use of creosoted ties will result in an average annual saving in excess of $500 per mile of track per year. This will mean an annual saving of $10,000 on the present main haulage now in use when all ties and switch ties are creosoted. In addition to the direct saving, the absence of wrecks and delays results in increased safety of operation and greater efficiency.

While these figures have been based largely on experience with creosoted ties on railroads, it is reasonable to expect that such ties will give even longer life in mines than on the surface because of the following factors:

Lighter maximum wheel load of 10,000 lbs. in mines compared with 35,000 lbs. on the railroads.

Uniform temperature in mines, with no freezing and thawing.

Freedom from weathering and checking caused by sun and rain.

Greater permanence of creosote due to absence of evaporation.

An niteresting service record of creosoted pine ties is being obtained by the Inland Steel Company at its Indianola mine in Allegheny County, Pa. In 1919, 4,800 lin. ft. of main haulage track was constructed, using approximately 2,600 sawed southern yellow pine 5 x 7—6 ft. ties ,treated by the Rueping process, with a net retention of 6 lbs. of 80-20 creosote coal-tar solution per cubic foot. These ties were ti-plated. A recent inspection of this piece of track reveals the fact that these creosoted pine ties are in good condition after 12 years of service.

It is evident that coal mining companies can also use creosoted timber profitably for other purposes than main haulage track construction. It is, in fact, already being used for a number of purposes in the mines operated by W. J. Rainey, Inc.

Creosoted wood block floors have been used for the past 12 years in the stalls of all stables, where untreated plank had formerly been used. The annual cost of these blocks is about one-half that of maintaining the untreated plank floors, and the blocks are much more satisfactory in every way.

Creosoted lagging is being used to some extent over steel I-beams to protect the roof in the main haulage.

Creosoted lumber has been used to some extent to repair porch sills and is proing very satisfactory for the purpose.

Records of the life of timber in mines have been difficult to obtain. Very few mines have kept detailed timber records in the past, and in some instances workings have been abandoned before the full life of the timber has been realized. This is in distinct opposition to the very careful records that the mines have kept of the life and service of certain types of hoisting cables, belts, etc. It is our conviction that time and money invested in an intensive study of timber in mines, with particular reference to sizes, species and durability (for both permanent and temporary use) will pay real dividends to the company making such investigation.

Washing Practice at Nellis Mines

(*Continued from page* 83)

the analysis of two series of samples taken during normal washing operations:

	0-4½-in. coal First series %	Second series %	Average %
Ash reduction	50.04	47.60	48.82
Recovery	89.18	90.10	89.64
Refuse	10.82	9.90	10.36
Qualitative efficiency	93.85	93.20	93.52
Quantitative efficiency	99.86	99.76	99.81
Bank loss	0.14	0.246	0.193

One of the most important considerations in the mechanical cleaning of coal is the character of the sink in the washed coal. It is obvious that a coal containing 2 percent of sink material which analyzes 35 percent is a better coal than one containing the same amount of sink containing 50 percent ash. It is the complete elimination of the very high ash material which, in general, raises the fusing point of the ash.

The Nellis 0 x 1½ in. nut and slack coal has been used almost entirely as boiler fuel in the Middletown plant. Prior to the installation of the washer the American Rolling Mill Company Combustion Department stated that a saving amounting to approximately 36 cents per ton in the cost of boiler fuel could be effected if the ash of the boiler coal could be reduced from an average of 13 percent to an average of 8 percent. I have understood this figure has been exceeded in actual practice as the boiler efficiency has been greatly increased and the use of one bank of boilers has been discontinued.

The 1½ in. x 4½ in. egg coal is used as fuel for the gas producers operating in the open-hearth furnaces of the Middletown plant. No definite figures were set up by the Combustion Department of the company as to the saving which would be effected by the use of better fuel in the gas producers. However, it was estimated that about 15 cents per ton would be saved.

The reduction in the percentage of sulphur in the gas coal by washing has been one of the real benefits derived from the Nellis plant, as it is very necessary to eliminate the sulphur almost completely during the manufacture of Armco iron in the open hearth furnace.

Prior to the installation of the washing the fusing point of the ash in Nellis coal was about 2,700 degrees. The fusing point of the washed coal is some 200 degrees higher or more than 2,900 degrees.

The reduction in ash in both the producer and the boiler coal, together with the raising of the fusing point of the ash, led to some experimental work being done towards the use of 0 x 3 in. coal as gas producer fuel. The experiments proved so successful that eventually all coal shipped to Middletown will be of the above size and will be dumped onto a common stock pile for use in either the gas producers or the boilers.

As stated above, the Nellis washer is a 200-ton per hour plant. During the period covered by the costs given below, the plant operated at about 60 percent of capacity. The costs, particularly power, are therefore higher than would obtain were the plant operating at near the normal tonnage. The demand charge during the period covered was about as high as the energy charge. The average power consumption was 1.59 kw. hr. per ton. With normal operation this should not exceed 1.25 kw. hr. per ton.

Operating costs are as follows, with no amortization or depreciation included for the period July 1-December 31, 1930:

Charge	Cents per ton mined	Cents per ton washed
Operating labor	1.27	1.73
Repair labor	0.58	0.79
Supplies	1.60	2.17
Power (demand charge)	1.50	2.04
Power (energy charge)	1.50	2.04
Total	6.45	8.77

The installation of the Nellis washer marked a somewhat new departure in the wet washing of fine coal by the Rheolaveur method. The successful operation of this plant has pointed the way to simplification of design with the consequent reduction in first cost and with but little if any sacrifice in efficiency or results.

No. 8 Cleaning Plant

of the Pittsburgh Terminal Coal Corporation using the Chance Sand Flotation Process

By Joseph Pursglove, Jr.

THE second bituminous coal clearing plant to use the sand flotation process was placed in operation on January 24, 1931, at the No. 8, or Coverdale Mine, of the Pittsburgh Terminal Coal Corporation, about 14 miles west of Pittsburgh.

Since the generic theory * of the sand flotation process and the factors † controlling the design of plants of this character have been hitherto outlined in previous papers on the subject, the present paper will be confined to a description of this particular plant and the operating results secured therefrom during the past three months.

The Original Coverdale Tipple

This plant was built in 1920 by Heyl and Patterson with circular picking tables for egg and lump coal and weigh basket equipment external to the dump. Coal was hoisted in self-dumping cages holding two mine cars each. Loading booms were provided for egg and lump only.

In 1929 this tipple was reconstructed by its original builders, the weigh baskets being dispensed with and the dump lowered some 20 feet in the head frame. At the same time a different type of shaker screen was installed, the circular picking-tables removed, and standard loading booms installed for the hand-picking of lump, egg and nut, together with belt conveyors for the separate loading of egg and slack. This revision of the original design increased the loading tracks to five in number and permitted a more flexible operation of the tipple.

The tipple as so remodeled was a representative modern shaft plant, permitting the greatest possible number of hand-pickers and at the same time allowing the loading of practically any desired combination of sizes by the employment of a remixing conveyor.

Character of Coal

The No. 8 mine is in the Pittsburgh bed which in this locality has all the average characteristics common to the Pittsburgh district. Preliminary float-and-sink tests made in 1930 showed that an operating density between 1.45 and 1.55 would serve to remove all objectionable refuse together with such high ash bone as would have a deleterious effect in the use of this coal either as a domestic fuel or for gas making purposes. The sulphur problem, while not a pressing one, however required some consideration. The raw coal contains about 1.7 percent total sulphur with an organic sulphur content averaging about .8 percent. Hence at least .9 percent pyritic sulphur is present and it is desirable that this be reduced as much as possible. These preliminary tests indicated that a shrinkage in weight of between 6 and 7 percent of the entire mine run would occur if an operating density of 1.45 were adopted.

After the usual inspection of various methods that might be employed for the preparation of this coal, it was decided to adopt the Chance sand flotation process for the preparation of all coal of a size large enough to justify washing, without including coal of such fineness as to cause moisture difficulties in the slack.

As has been previously noted ‡ in The Mining Congress Journal it is not always necessary to wash the fine coal passing a ¼" or 3/16" hole screen if the coarser sizes are properly reduced in ash, such by-passing of the raw fines eliminating both the sludge and the moisture problem. It was felt that the coal produced by the No. 8 mine would lend itself to this procedure and therefore washing could be restricted to the coal coarser than ⅝", ⅜" or ¼", depending on the markets into which the slack coal was to be placed. This decision at once eliminated the necessity of any large water clarification or recovery installation as it was evident that the Mt. Union practice of eliminating sludge by means of the makeup water overflow, assisted by the various sump sludge discharges, would suffice. The plant design therefore followed these general principles and the plant as operated today is running along these lines.

Present Coal Cleaning Plant

Lump Coal Treatment

The design adopted utilizes without material change the existing lump coal shakers. The raw coal is dumped from the cages on a pair of 60" apron feeders, each feeder discharging to a 6' lump shaker. The 1¼" slack is removed on the first section of these shakers and passes directly to a 6' wide Heyl and Patterson Jouncer screen, this screen being adapted to take out ⅝", ½" or ⅜" slack. The coal remaining on the lump shaker then passes over a screen fitted with 4½" perforations which removes the 4½" x 1¼" raw coal. The remaining lump is then collected on a 6' apron loading boom, the lump refuse being removed by six hand-pickers, with refuse disposal to a 12" drag chain conveyor, this latter delivering into a 25-ton storage bin. All of this equipment, including the Jouncer screens, was installed in the tipple prior to the time of conversion to a wet cleaning plant.

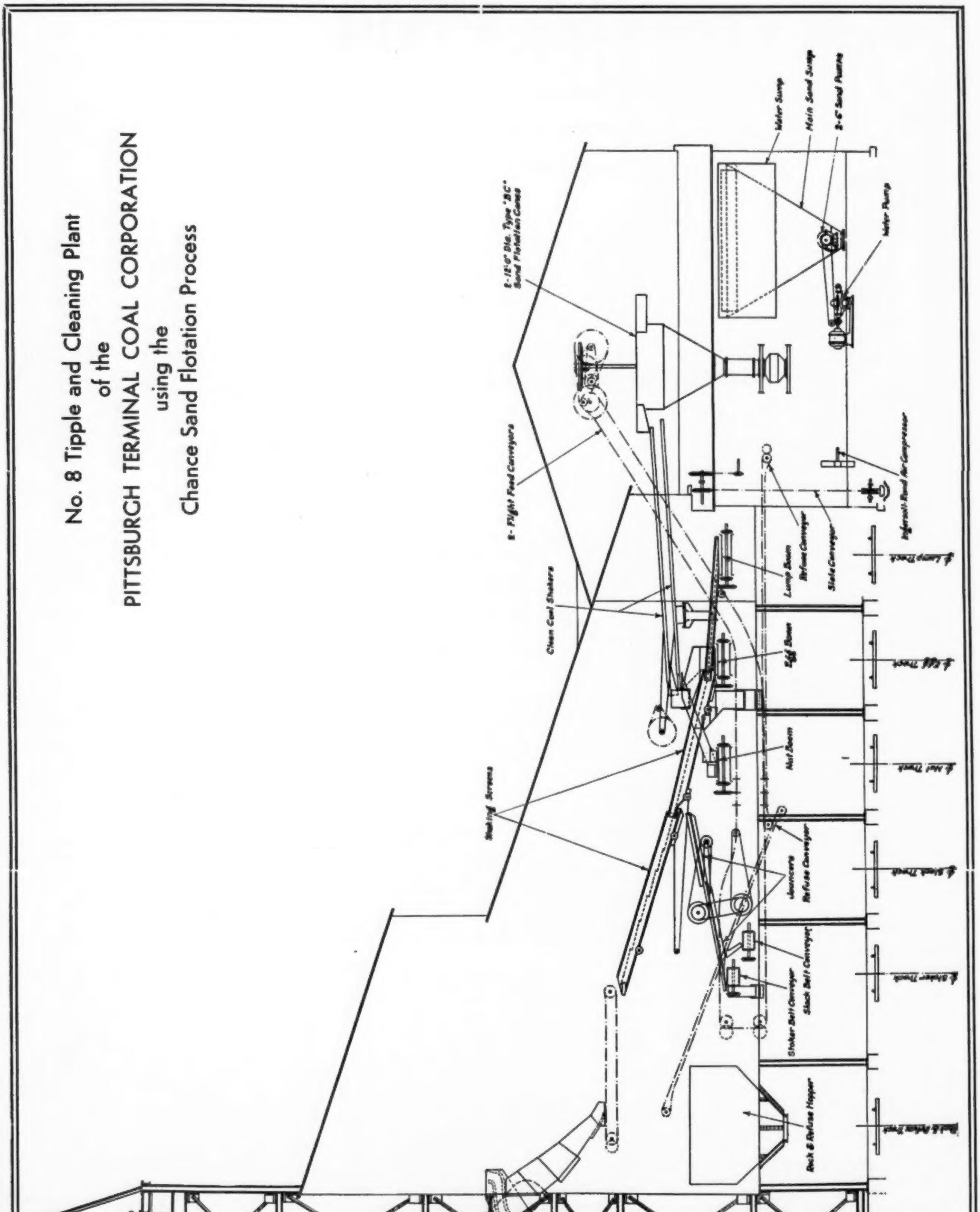
Cleaning and Screening 4½" Undersize

The 1¼" undersize made by the Jouncer screens passes into a pair of 12' x 24" flight conveyors, which later pick up the 4½" x 1¼" undersize removed by the lump shakers. These conveyors elevate the raw coal and each one delivers into a 12' Chance sand flotation cone. These cones follow the same general design of the Mt. Union units but are of larger diameter and are fitted with the standard 24" throat refuse valves used in anthracite practice. The clean coal floating in the top of the fluid mass discharges from each cone to a 6'-0" x 34'-6" desanding, dewatering and sizing shaker, these shakers having top decks of 6'-0" width and bottom decks of 7'-0" width. The clean coal is sized into 4½" x 2" egg, 2" x 1¼" nut and 1¼" pea coal, the first two sizes passing to loading booms and the pea being carried back by the return strand of each 12" x 24" feed conveyor to the present pea coal belt loading conveyor. One section, 3'-0" long, of Hendrick drainage screen, fitted in each of these return strands, serves

* T. M. Chance: a New Method of Separating Materials of Different Specific Gravities, Trans. A. I. M. E. (1918) 59, 263.

† T. M. Chance: The Sand Flotation Process as applied to the Washing of Coal, Mining Congress Journal (1928), Sept.

‡ T. M. Chance: Mining Congress Journal (1928, Sept.), page 727.


No. 8 Tipple and Cleaning Plant
of the
PITTSBURGH TERMINAL COAL CORPORATION
using the
Chance Sand Flotation Process
Shaking Screens
Clean Coal Shakers
Lump Boom
Egg Boom
Nut Boom
Refuse Conveyor
Slate Conveyor
Rock & Refuse Hopper
Water Sump
Main Sand Sump
Water Pump

to remove any extraneous water from the pea coal that may be retained after discharge from the shaking screens.

The refuse is discharged from the cones in the usual manner through a refuse draw of the free-discharge type and after desanding and dewatering is delivered into an 8" x 16" refuse conveyor which in turn delivers into the existing 12" drag chain used for the lump picking-table refuse.

The raw slack, the size of which is controlled by the mesh that may be in use on the Jouncer screens, is delivered to the existing 30" slack belt conveyor and carried out to the loading chute.

The combined loading of the various sizes is made possible by an 8" x 30" scraper conveyor located in a structure adjacent to the ends of the loading booms, the pea belt conveyor and the slack belt conveyor. When mixed sizes are to be loaded, the various booms are elevated so as to discharge directly into this scraper line and the mixed coal is then conveyed to a point over the boom from which the coal is to be finally loaded.

Sand Return and Water Pumps

The mixed sand and water reclaimed by the sizing shakers and from the refuse shakers is carried to a common sand sump, fitted with an external water sump. The comparatively clear water overflowing this sand sump passes to this water sump and is then recirculated by a single stage centrifugal pump. The lower portion of the sand sump is provided with two slow speed sand return pumps, delivering directly to each 12' separating cone. The sand sump is provided with sludge removal devices and an efficient separating device, permitting direct recirclation of the sand from the lower portion thereof and the water from the upper portion thereof. Makeup sand is stored in a concrete bin adjoining the washery and served for its full length by a half-yard grab bucket motor operated on a trolley located above the bin and permitting unloading of sand cars spotted on the lump loading track. All pumps are of types developed by the Morris Machine Works.

Slate Gate Control

Slate gates are operated by compressed air from an Ingersoll-Rand single stage compressor, the control being completely automatic. This control is of the electrical type, including definite time relays of stock type built by the General Electric Company and solenoid operated piston valves constructed by the Ruggles-Klingemann Manufacturing Company. The entire timing is under the control of the washery operator and any desired interval of opening the slate gates can be secured within the range of the timers up to a maximum delay of 36 minutes. This automatic control gear has been highly successful, both speeding up the refuse withdrawal and resulting in mathematical accuracy in the relative timing of the various components of the cycle. It was feared when this equipment was designed that certain delays might be occasioned by failure of the slate gates to completely close or open due to the presence of large pieces of slate, etc., but such occurrences are so rare as to be of no importance.

Recovery of Makeup Water

The waste water discharged from the washery averages about 200 gallons per minute. This water consists wholly of the makeup water which is supplied to the plant, and is introduced through the last of the clean coal shaker sprays so as to wash the coal with perfectly clear water before final discharge.

The waste water, carrying a percentage of sludge approximately the same as the dilute water returned by the main circulating water pump in the washery, passes through an 18" sewer to a waste pond, one wall of which is the existing refuse bank adjacent to the No. 8 shaft. This water rises to a given level and then filters through the waste bank and passes into a collecting basin from which it is returned to the washery as clear makeup water. An idea may be gathered as to the small amount of silt passing out with this 200 g.p.m. of washery waste water, and also as to the efficiency of such a refuse bank as a filter medium, from the fact that the water recovered from the collecting basin is perfectly clear pure water, such as one would drink if he were not aware of the possible source of pollution.

This remarkable clarification result is an evidence of the extremely efficient manner in which the sand flotation process removes fireclay and friable shales, these materials passing down to the refuse discharge gates and out of the unit before they have time to either dissolve or be broken down into small fragments. With washers retaining the refuse for any length of time in an agitated condition such a water clarification result would be difficult to obtain because of the colloidial material produced by prolonged attrition of such fireclays or shales.

RAW COAL AND REFUSE—BY PITTSBURGH TERM. COAL CORP.

Size	Ash	Moisture
4½" x 2" Egg	6.82	2.19 R. O. M.
2" x 1¼" Nut	8.18	
1¼" x ⅜" Pea	9.00	
4½" x ⅜" Refuse	54.00	

WASHED COAL—BY COKE PRODUCER

Size	Ash	Moisture as unloaded
4½" x 2" Egg	4.77	1.6
2" x ¼" Nut	5.48	2.6
1¼" x ⅜" Pea	5.80	3.4

COKE—BY COKE PRODUCER

Mixture Coked—80 percent Pittsburgh Terminal 4½" x 2" Egg @ 4.77 percent Avg. Ash.
20 percent Pocahontas 1" Slack @ 5.13 percent Avg. Ash.

Average Ash in Coke—7.78 percent.

Attention is particularly called to the average moisture results given in the second table above, and also that these low ash coals have been produced without increasing the forecast washery losses.

Daily Float-and-Sink Tests

Float-and-sink control tests with a mixture of CCl_4 and gasoline are run on each car of washed coal loaded and every hour on the refuse. Operating density is held at 1.45 sp. gr. Daily averages are as follows:

Material	Float	Sink
Washed Egg	100.0	0.0
Washed Nut	99.5	0.5
Washed Pea	99.5	0.5
Refuse	1.0	99.0

Practically all sink material in washed coal samples will float at a specific gravity of 1.50 or 1.55 and no pieces of float in the 4½" x ⅜" refuse are ever larger than 1½" size, consisting always of chippings that will generally sink at 1.40 specific gravity.

An average float-and-sink analysis on representative samples of Egg, Nut and Pea shipped several hundred miles to another laboratory for check test is given in the accompanying table.

Material	Float @ 1.45	Sink 1.45 Float 1.50	Sink 1.50 Float 1.60	Sink 1.60	Total Sink
4½" x 2" Egg	99.8	..	..	.20	.20*
2" x ¼" Nut	99.53	.39	.02	.06	.47
1¼" x ⅜" Pea	99.19	.54	.04	.23	.81

* All Sink 1" x ⅜" chippings.

Operating Results

Daily Tonnage and Washery Delays

The plant was designed for a maximum hourly hoist of 800 tons, with 480 tons of 4½" x ⅜" coal passing to the washery. The forecast output was 4,000 tons per average 8-hour day.

The average daily shipped tonnage for the first 25 working days after the plant started on January 24, 1931, was in excess of 4,000 tons, with a minimum day's production of 2,560 tons—on a day that that a shaft accident suspended hoisting operations.

During the month of April, the total delay due to the washery and/or its auxiliaries was 5 minutes, on a tonnage in excess of 90,000 tons. I believe that this record has never been surpassed by any coal or other metallurgical plant of any size or type.

Ash and Moisture Analyses

Average analyses of the raw coal, refuse, washed coal and coked product are as follows:

Sand Consumption

Average sand consumption is 3 one-half yard buckets per shift or 3,600 pounds, averaging .9 pounds per ton of total production or 1.7 pounds per ton of washed coal shipped. This is the lowest sand consumption that has ever been recorded in a Chance sand flotation plant of any type.

Power Consumption

WASHERY POWER TEST OVER 2 WEEKS PERIOD

	H. P.	Power Factor
Installed load	350	..
Peak load	322	.80
Minimum load	236	.65
Average load	279	.78

LOAD FACTOR

	Percent
Peak	92
Minimum	68
Average	80

K. W. H. PER TON/HOUR WASHED COAL

	K. W. H.
Peak @ 600 T/H	.4
Average @ 350 T/H	.6

Force Account

The total washery and tipple force account is as follows:

Foreman	1
Tipple operator	1
Cone runner	1
Coal inspector	1
Dumper	1
Lump pickers	6
Railroad car handlers	6
Washery utility	1
Refuse disposal	2
Machinery attendant	1
Maintenance (night)	1
Watchman (night)	1
Total	23

This force account of 23 men, of which 2 are allocated to washery and one to final coal and refuse inspection, includes all of the plant operatives required between the headframe dump of the raw coal and the finished product going to market and the refuse bank. All the coal larger than slack is now prepared with nine less men than were formerly required for producing the same total production when there was no cleaning of the pea coal and practically no cleaning of nut coal.

Maintenance

Up to this time in excess of 250,000 tons total production have been shipped with no major renewals of any kind. The washery maintenance during this period has included:

Coal shakers	—2—	⅝" round hole steel screen plates
Coal shakers	—4—	3/32" round hole 7'-0" bronze screen plates
Refuse shakers	—6—	3/32" round hole 4'-0" bronze screen plates
6" sand pump	—1—	Cast iron throat wear rings
6" sand pump	—1—	Cast iron throat wear rings
4" sand pump	—1—	Cast iron throat wear rings
Air compressor	—1—	Belt (accidentally burned out)

No sand pump impellers or casings have been renewed, the Diamite alloy used for these units coupled with the comparatively pure water, now recirculated from the collecting basin, standing the wear in excellent shape.

The total cost of the above replacements amounts to $362, or 1.248 mills per ton of total production.

Operating Cost

While a three months' operating campaign would not ordinarily justify an accurate estimate of operating cost, still I believe that the costs at No. 8 for washing and clean coal sizing will correspond to the following units per ton shipped through the washery:

Labor	.02 man hours
Sand, all maintenance and supplies	1.0 cent
Power	.6 K. W. H.
Capital invested for washery	$100,000

Shaker Screens

The shaking screens are of the most simple design and have operated up to the present time without the breakage of a single driving arm or hanger board. These shakers are of wood construction with flexible arms, the coal shakers operating at 180 and the slate shakers at 200 r. p. m. I believe that the remarkable performance of these screens is due to:

(a) Inherently correct design, eliminating practically the usual eccentricity of the driving arm connections.

(b) Light weight and balanced construction.

(c) Excellent erection on the part of the contractor's force.

Capital Cost of Plant

A duplicate of the No. 8 cleaning plant, including all changes to a similar tipple structure, makeup water supply, all auxiliaries, and the same engineering expense, could be installed under present conditions for less than $100,000. The No. 8 plant is of steel construction throughout and operates smoothly and without vibration—the steel sections and bracing employed being of greater than ordinary weight. Such a cost would apply to a structure of similarly sound construction.

Conclusion

The entire No. 8 conversion was designed by H. M. Chance and Company, with the writer in charge of field correlation, design of makeup water supply system, etc. The plant was constructed by Heyl and Patterson as general contractors, the structural steel and building being furnished to them by the Pittsburgh Engineering, Foundry & Construction Company, the cones, sumps and chutes by the McCarter Iron Works, pumps by the Morris Machine Works, motors and controls by General Electric, switches by Westinghouse, Texrope drives (which latter are used on all shakers and the cone drive) by Allis Chalmers, and miscellaneous conveying equipment by Link-Belt.

I feel that, notwithstanding the comparatively short time the coal produced by this plant has been on the market, consumers are rapidly coming to appreciate its value—not only as a gas and domestic coke fuel—but as a generally desirable high volatile coal. That such views are held by those purchasing the coal is best evidenced by the fact that not one idle day since the first of April has occurred by reason of lack of orders.

I wish to express my appreciation of the assistance of Mr. Samuel Pursglove, president, and my associates in the Pittsburgh Terminal Coal Corporation, in securing the data presented in this paper and further for their sincere cooperation in making this plant the outstanding success that I believe it to be.

Treating Machine Bits

(*Continued from page* 74)

the prepared bit to the cutting machine at the face are listed below, together with the difference in cost experienced:

	Decrease	Increase
New bit steel	.0006	
Heating and sharpening labor	.0033	
Special preparation labor		.0033
Supplies		.0008
Repairs to equipment	Negligible	
Delivery to face labor	.0006	
Investment charges on shop equipment	No change	
	.0048	.0041

The figures, while insignificant in themselves, show that the saving in bit steel balances the extra material necessary in the special process, the saving in heating and sharpening labor balances the cost of the special labor, and the saving in delivery cost makes the total cost slightly less than before.

The ultimate effect of the use of the bits on the various items entering into the cost of cutting coal is shown below:

	Savings
Cutting labor	9%
Machine repairs	36%
Oil	No change
Power	21%
Bits	No change (as noted above)
Machine investment	Not considered

Including all factors the saving was 15 percent.

It has been noted that the floating dust in the air around a machine using the special bit was far less than under the former system. Thus the hazard attending the cutting operation is greatly reduced.

Conclusions

The cost of cutting coal may be decreased within certain limits by the use of improved machine bits.

This is possible (1) through the elimination of delays to the cutting unit caused by frequent bit renewals and waiting for deliveries of bits, (2) through the saving in power consumed and the resulting lower cost of machine repairs, and (3) through the increase in the cutting capacity of a given machine, thus requiring less investment in machinery to cut a given tonnage.

The difference in cost of any two types of bits delivered to the machine, balanced against the difference in actual cutting costs when using the two types of bits, should determine the relative values.

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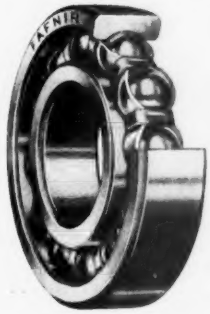

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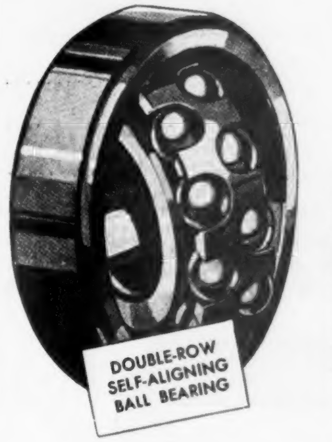

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Steel Lined Sheaves, Electric Car Retarders, Coal Lowering Spirals, Handling and Weighing Equipment, Complete Tipples.

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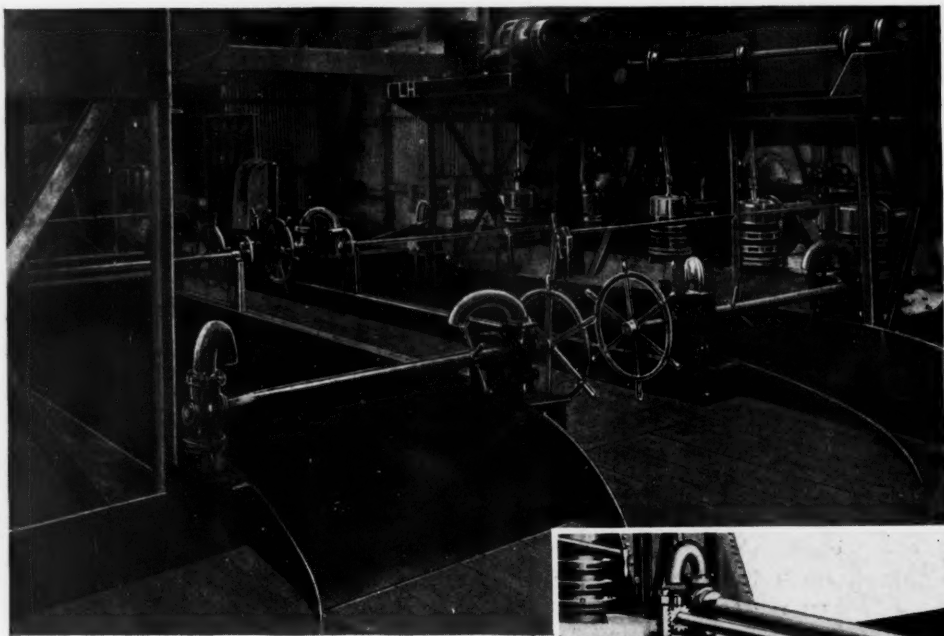
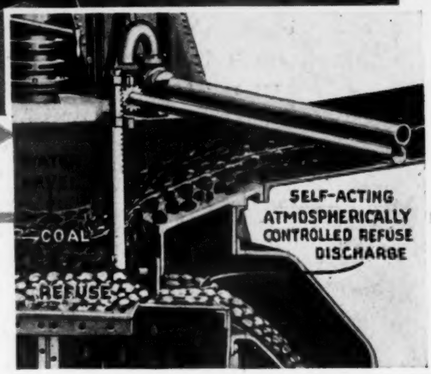


Outlet gate for heavy refuse at the entry side of the Link-Belt—Simon-Carves Wash Box. Self-acting and atmospherically controlled. There is a similar gate for the smaller refuse at the opposite side of the box.



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